Metallurgist

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NOTICE

Translated from Metallurg, No. 9, inside front cover,
September, 1960

Esteemed Reader! We remind you that in the very near future subscriptions will be taken for the monthly, popular production and technical journal, "Metallurg" for 1961.

The resolutions of the July Plenum of the Central Committee of the Communist Party of the Soviet Union have aroused a powerful surge of creative enthusiasm throughout the entire country. Everywhere the movement is expanding for the introduction of the most advanced methods of work, for mechanization and automation of the productive processes. The Soviet metallurgists set out in the vanguard for the early fulfillment of the Seven-Year Plan, for the technological rearmament of production.

Under these conditions the role of technical information, the exchange of production experience between related sections, shops, and enterprises, acquires still greater importance.

The journal "Metallurg" sets as its goal the extensive elucidation and dissemination of the operating experience of metallurgical enterprises, of the introduction and application of new techniques and technology of mechanization and automation of production.

On the pages of the journal "Metallurg" in 1961 you will read about the achievements of innovators and efficiency experts of metallurgical plants, about the introduction of complex automation and mechanization of technological processes, about highly productive and economic processes, new machines and units for ferrous metallurgy. The journal gives qualified consultations on problems of interest to the readers.

The wide participation in the work of the journal "Metallurg" of all metallurgists will permit us to make the journal a true bearer and propagandist of technological progress. Send your articles, your wishes to the office of the editor, tell about the experience of introducing new technique and advanced technology, about making the production process more efficient at your sections.

Each skilled worker, engineer, qualified worker should subscribe to the journal "Metallurg" where he will be able to find the answer to many questions of interest to him, to learn about the work of the most advanced enterprises.

The journal "Metallurg" is a valuable publication not only for workers in metallurgical and machine-building plants. The journal can help the students of metallurgical institutes and technical schools to master successfully their specialty, to correlate theoretical knowledge with the operating experience of the leading enterprises.

In order to avoid an interruption in your receiving of the journal, fill out the subscription form well in advance.

Subscriptions are being taken by the municipal and regional departments of the Union Press, by offices and departments of communication, and by public authorized agents.

AT THE EXHIBITION OF ATTAINMENTS OF THE NATIONAL ECONOMY OF THE USSR

Translated from Metallurg 1960, No. 9, pp. 1-2, September, 1960

In connection with the fact that not all metallurgists can visit the Exhibition of Attainments of the National Economy to become acquainted with the exhibited samples of equipment and with new technological processes, the journal "Metallurgist" and the ferrous metallurgy section of the Committee of the Council of the Exhibition have decided to print a short description of the exhibits presenting the greatest interest for workers in ferrous metallurgy.

In December 1959 the Central Committee of the Communist Party of the Soviet Union and the USSR Council of Ministers adopted a resolution "On improving the work of the Exhibition of Attainments of the National Economy of the USSR and elevating its role in the matter of introducing the attainments of science, technique and advanced experience into the national economy."

This resolution, directed toward a fundamental reorientation of the work of the Exhibition, puts the question of its operations in a completely new way and elevates the role of the Exhibition in the matter of introducing attainments of science, technique and advanced experience.

The Central Committee of the Communist Party of the Soviet Union and the USSR Council of Ministers have pointed out that the principal task of the Exhibition is the active propagation and introduction into industry, transportation, agriculture and construction of the attainments of science and technique, progressive technology and inventions promoting technical progress.

The organization of specialized thematic exhibitions on current problems in the development of industry, agriculture, construction and transportation is acknowledged to be one of the most important forms in the introduction of the attainments of science, technique and advanced experience. They should be of short duration and bear the character of reports of the councils of the national economy, enterprises, construction, collective farms and state farms on the level attained in production, its economics and the technical level of machinery, equipment, instruments, materials and other production.

During the thematic exhibitions seminars and scientific-technical conferences will be held with the participation of production workers, scientific-research and planning-design organizations, as well as scholars, inventors, efficiency workers and innovators of production.

According to the results of the holding of these exhibitions the Council of the Exhibition jointly with the State Committees, Ministries, Departments, the Councils of Ministers of the union republics and other organizations, is required to submit to the USSR State Planning Commission, and in necessary cases jointly with the USSR State Planning Commission and the USSR Council of Ministers, suggestions regarding the introduction of new technique and advanced experience, as well as the removal from production of obsolete machines, instruments and parts.

The selection of new exhibits and participants in the Exhibition will be made throughout the year to the extent of the creation of specimens of new technique, advanced methods and the organization of production, and not once a year as has been done previously.

In accordance with the decision a Council of the Exhibition has been organized for the management of

the Exhibition and determination of basic directions in the content of its work. Sections have been created in the Committee of this Council which review the thematic and thematic-expositional plans, select the best specimens and work out plans for the conduct of special thematic and mobile exhibitions, etc.

A section has been created for the determination of the basic directions and content of the ferrous metallurgy exposition; it includes directors of institutes and metallurgical enterprises and specialists of the State Planning Commission and State Committees.

In the second half of 1960 a thematic exhibition will be organized in the halls of the ferrous metallurgy division on the production and use of economic cast iron profiles in connection with the fact that individual cast iron profiles, especially the curved, are not being introduced sufficiently into machine-tool building and construction. The efficiency of the use of curved profiles in the various branches of industry will be extensively demonstrated at the Exhibition, as well as the possibility of manufacturing profiles not yet mastered but necessary for the national economy.

Workers from factories, councils of the national economy and institutes will be invited to the thematic exhibition on cast iron profiles to study them and develop suggestions.

Along with the thematic exhibition various types of equipment and processes will be demonstrated in the exhibition halls which are recommended for introduction into blast, open hearth and rolled iron production, as well as the coke and chemicals industry.

This year, at the permanent exhibit of the ferrous metals industry, located in three halls of the "Machinery" pavilion, specimens have been arranged which reflect the present level of development of the metallurgical industry. As regards ore concentration, concentrating and agglomerating plants are being displayed, as well as a working model of an agglomeration machine, a rotary roasting for limestone in the stock section of an agglomerate plant, a drum magnetic separator, a magnitograph and specimens of blast agglomerate with a different degree of basicity from that of iron ores native to the USSR.

Modern blast furnace production is represented by a working model of a blast furnace capacity of 1513 m^3 with an illuminated diagram of the automatic control of the working process.

The Magnitogorsk Metallurgical Combine is sponsoring a method they have mastered of conveying the charge to the skip hoist of a blast furnace, a rotating gutter for the output of cast iron and a two-layer specimen of a nozzle for the delivery of highly heated blast. The Dnepropetrovsk Council of National Economy has presented samples of greased nozzles mastered at the Petrovskii Works. Experience in the use of natural gas in blast furnace production will be disclosed by the Zaporozhsk and Dnepropetrovsk councils of national economy.

The outfitting of the steel smelting section has been given special attention. Models of smelting aggregates have been created on the basis of drawings of operating modern metallurgical aggregates. The State All-Union Design and Planning Institute of the Ministry of Ferrous Metallurgy is exhibiting a working model of a converter with top feed of oxygen. According to the Central Scientific Research Institute of Ferrous Metallurgy, converter steel produced with the use of oxygen at the Petrovskii Works is not inferior in mechanical properties to open hearth steel.

The "Elektropech'" trust has shown a working model of an 80-ton electric furnace with electromagnetic mixing.

A working model of equipment for smelting steel under a vacuum offers great interest; this method has been mastered by the Ural Heavy Machinery Plant (UZTM). Observation of the casting of ingots is done by television, and the weight of an ingot which can be cast at the UZTM is 120 tons. According to the data of this plant the structure of the metal and the working conditions have been improved and the annual output has been increased.

The Central Scientific Research Institute of Ferrous Metallurgy and the State Institute for the Design and Planning of Steel Structures are exhibiting a working model of a four-groove pouring machine of the Stalinsk plant for continuous pouring of steel, put in service in July, 1960. The pouring process is automated and completely mechanized. Observations are made by television.

The Kuznetsk Metallurgical Combine is displaying this year a working model of remote control stoppers in the pouring of steel.

A modern standard open hearth works with 500 ton furnaces, the largest in Europe, and a 1300 ton mixer is being exhibited. A new scheme of automatic control and regulation of the heating conditions has been put in operation.

Rolling mill production is displayed by a working model of a 1150 blooming mill with fully automatic control; the model was built according to drawings of the Ural Heavy Machinery Plant on a scale of 1:20. It demonstrates visually the entire process of rolling, from heating of the ingot in the heating blocks to the stamping of slabs and blooms. The model characterizes modern mills with the presence of two-ingot feed, high-speed pressure equipment, mechanization of removal of large scrap, unreduced drive of the manipulators and mechanized stamping.

An automatic guiding apparatus for the transfer of the oval to the ferrous line of the wire-drawing mill is exhibited by the Dnepropetrovsk Council of National Economy. According to the data we have, this apparatus increased the productivity of the mill and eliminated manual labor.

The Central Laboratory of Automation of the RSFSR Ministry of Construction displayed a contactless thickness meter TPU-495, which automatically measures strip thickness from 0.03 to 1 mm with a width up to 50 mm, and photoelectric pyrometer FÉP-3 for measuring temperature in the rolling process.

The section contains exhibits of economic, periodical curved profiles and new forms of rolled iron mastered in recent years.

Tube production was represented by a working model of a roller bearing mill for rolling thick-walled pipe (with a wall thichness of 0.2-0.8 mm). The Pipe Institute of the Dnepropetrovsk Council of National Economy is demonstrating working models of apparatus for electrically polishing the inside and outside surfaces of pipe. These apparatus completely replace manual polishing and improve the mechanical and physical properties of the pipe.

The Khartsyzsk Tube Plant of the Stalinsk oblast is exhibiting models of tube-welding aggregates for welding with outside and inner seams of pipe with diameter greater than 1 meter.

Metalworking production is exhibited with representative specimens of various types of screens made of alloyed metal, brass and carbon steel. The exposition reflects modern processes and mechanization of the manufacture of screens.

Samples of metalworking production are shown on stands: bolts, nuts, cables of various design, wire and fastening devices, as well as modern methods of manufacturing them.

Fourteen specialists of the Ural Heavy Machinery Plant were awarded medals and valuable prizes, and the plant a first degree certificate, for participation in exhibiting vacuum processing of steel in the exhibition hall in 1959. In the entire division of industry and transportation, certificates were awarded to 596 enterprises and institutes and 5408 men were awarded medals and valuable prizes.

The scientific research and planning technological institutes, especially the Central Scientific Research Institute of Ferrous Metallurgy, the State Institute for the Design and Planning of Metallurgical Plants, the State All-Union Design and Planning Institute fo the Ministry of Ferrous Metallurgy and the All-Union Scientific Research Institute of Heavy Machinery, and advanced metallurgical plants and specialists should take an active part in the work of the Exhibition, and will thus in a most excellent way fulfill the resolution of the Central Committee of the Communist Party of the Soviet Union and the USSR Council of Ministers regarding elevation of the role of the Exhibition of Attainments of the National Economy (VDNKh) in the matter of introducing technique and advanced experience into the national economy of the USSR.

Many specialists from different branches of industry go to the Exhibition to study advanced experience. It is important that they obtain the necessary material on new technique and the work experience of innovators, so that all that is best and newest presented at the Exhibition by leading enterprises, institutes and inventors becomes the property of all branches of ferrous metallurgy, of the entire national economy.

I. B. Zadzishek

PRODUCTION OF CAST IRON ON LOW-BASICITY SLAGS

M. L. Lavrent'ev, I. V. Kotel'nikov, F. P. Tarasov,

and V. P. Tarasov

Il'ich Works

Translated from Metallurg 1960, No. 9, pp. 3-6, September, 1960

Production of low-sulfur cast iron in blast furnaces entails a reduction in furnace capacity and an increased coke consumption. This is particularly the case at the Southern plants which operate on high-sulfur Donetskii coke and low-alumina ores. In order to produce low-sulfur cast iron, blast furnaces as a rule operate on high-basicity slags ($CaO:SiO_2$) = 1.35-1.45. In order to improve process and economic indices a process for operating on low basicity slags to produce low sulfur cast iron was developed and put into operation on the No. 1 blast furnace at the Il'ich Works.

When choosing the smelting conditions it was reasoned that: a small decrease in slag basicity should not bring about any substantial decreases in crucible temperature, the conditions favoring reduction of silicon into pig iron will improve, the quantity of slag will decrease and the charge column porosity will increase. All these phenomena should give conditions favorable to increased furnace capacity.

The No. 1 furnace has been designed for operation with a top pressure of 1.5 atm. During the test period (March-May 1959) operating performance on cast iron was better.

The charge consisted of raw Krivoi Rog ore, Chiaturi Grade IV ore, Elenovsk limestone and Donetskii coke. Average composition of the charge during the test period is shown in Table 1.

The coke contained 2.27% moisture, 9.71% ash, and 1.74% sulfur. Its average tumbling index was 341kg. During the test period there were no major deviations from the average composition. However, because of the absence of appropriate equipment, the degree of blending of the ore was not adequate and the slag composition and furnace temperature fluctuated somewhat; this had the effect of lowering furnace productivity. Among the various slags, fluctuation of iron was up to ±3%. A characteristic charge property at that time was the low iron and alumina content in the ore, and the high content of insoluble residue.

Raw-material and fuel consumption factors are given in Table 2.

As can be seen from Table 2 iron-ore, coke and limestone consumption varied largely because of fluctuation in the metallic charge. Because of the low iron content in the ore, the output of slag was 950-1150 kg / ton of pig iron. In the period of March to May the furnace operated smoothly with a top pressure of 1.34-1.40 atm, yet when hot pig irons were smelted (silicon content higher than 3.25%) charge sliding was noted. Basic process performance data for this period are given in Table 3.

During the tests the furnace was operated with a blast temperature of 950° and near normal air humidity. The weight of the coke charge was 4 and 8 tons; charging sequence was two charges of ore-ore-coke-coke↓ and three charges of coke-ore-ore-coke↓. Charge level was 1.5 meters.

Such a charging scheme and a high top-blast pressure insured an optimum gas-flow distribution along the furnace diameter throughout the charge, as shown by the CO_2 concentration which was 7-8% at the wall and 3-4% at the center. Maximum CO_2 concentration (14-16%) occurred 0.8-1.3 m from the wall. Such a gas distribution insured a strong and stable heating of the furnace center and the crucible; the latter is a necessary condition for

an even furnace operation and the obtaining of pig iron with a low sulfur content while coke consumption is at a minimum. The top gases analyzed 9-10.5% CO_2, 30.5-32.3% CO and 1.5-2.2% H_2.

TABLE 1

Chemical Analyses of Raw Materials

Material	Analysis, %								
	Moisture	Fe	SiO_2	Al_2O_3	CaO	MgO	MnO	S	P
Iron ore	3.0	51.60	22.09	2.10	0.44	0.47	0.06 Mn	0.015	0.037
Manganese ore	13.7	2.19	36.83	6.59	3.02	0.78	25.04	0.200	0.179
Limestone	—	—	1.50	1.03	53.50	1.39	—	—	—
Metal additions	—	85	—	—	—	—	—	—	—
Coke ash	—	23.42	35.64	21.55	4.33	1.10	0.71	0.32	0.25

TABLE 2

Consumption of Raw Material and Fuel

Month	Material consumption per kg of pig iron				
	Coke (dry)	Iron ore	Manganese ore	Metal addition	Limestone
March	1090	1855	49	84	989
April	1032	1648	44	161	903
May	1047	1660	45	134	902
Average for the period	1056	1722	46	126	931

It is seen in Table 3 that the average "K.i.p.o."* of the furnace was 0.881 for the three months, which is a substantial achievement for this type of charge condition.

One of the more significant factors which resulted in the high furnace productivity was the slag regime. Because low-alumina ores were smelted, the furnace operated with unstable slags. Accordingly, it was necessary to find a slag basicity which would give a sufficiently fluid slag under wide fluctuations in composition and temperature. At the same time this slag should have a high tendency to remove sulfur.

In March 1959 the furnace began operating with a slag of 1.20-1.30 basicity. The viscosity of these slags did not exceed 6-7 poises at 1500°.

Mid-month analyses of smelting products are shown in Table 4. Average basicity varied between 1.16-1.35. Al_2O_3 content in the slag varied between 5.66-9.93% and MgO between 0.44-2.45%. During the experimental period pig iron containing 2.0-3.9% silicon was produced, whereby 65% of the results were below 3.25% and 35% in the range of 3.25-3.90%.

The experiments have enabled establishing the relationship between sulfur content in the pig iron and slag basicity at various silicon contents in the pig iron. This relationship is shown in Fig. 1; there the results of 522 pig-iron and lower slag tappings have been reproduced.

From Fig. 1 it follows that at a silicon content of 2.5-3.0% the sulfur content in the pig iron increases with an increase in basicity. When the silicon content is 3.01-3.25% there is very little dependence and the curve is almost parallel to the abscissa. When the silicon content in the pig iron is increased still further the sulfur content decreases with increasing basicity. Accordingly, in order to obtain low-sulfur (below 0.030% S) cast iron, slags should be the more basic the higher the silicon content in the pig iron.

In order to obtain an even clearer picture of the interdependence of slag basicity and the silicon content in the pig iron all results were divided into two groups according to the silicon content, i.e., one group 2-3.25% silicon, and the other 3.26-3.9%. The same relationship is shown in Fig. 2 for both these groups.

From Fig. 2 it is seen that, when smelting pig iron containing 2.0-3.25% silicon, the sulfur in pig iron increases from 0.022% to 0.044% as the average slag basicity increased from 1.22 to 1.27%. This may be explained by the increase in slag viscosity as basicity increased.

*Ratio of the working volume to the daily output.

TABLE 3

Blast-furnace Operation

Index	March	April	May	Average for the period
"K.i.p.o." m^3/daily ton	0.892	0.850	0.903	0.881
Downtime per month, hr-min.	3-47	4-43	7-35	5-22
Smelting rate based on wet coke, ton/m^3 day	1.086	1.080	1.031	1.066
Blast rate, m^3/min	2467	2461	2400	2442
Hot blast temperature °C	895	886	901	894
Blast moisture gm/m^3	18	17	13	16
Blast pressure, atm	2.45	2.36	2.40	2.40
Pressure at furnace top, atm	1.40	1.37	1.34	1.37
Exhaust-gases temperature, °C	269	302	291	287
Dust losses with gases kg/ton pig iron	48	46	49	48
Ore to coke ratio, ton/ton of moist coke	1.66	1.56	1.54	1.59

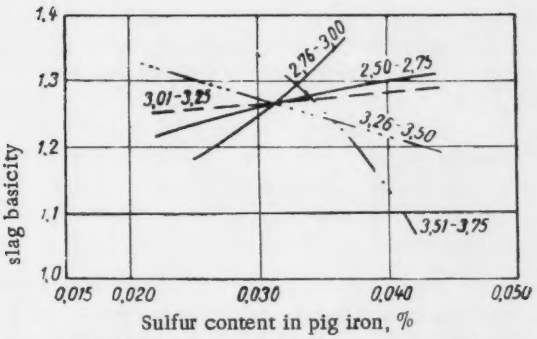

Fig. 1. Dependence of the sulfur content in pig iron on slag basicity at varying degrees of crucible heatup. Numbers on the curves denote ranges of silicon content in the pig iron.

The lime content in slag rose from an average of 48.7 to 49.19%, and silica content dropped from 39.87 to 39%. Concurrently the viscosity increase was 2 poises, and the viscosity of the slag which had a basicity of 1.27 reached 5 poises at a lower slag temperature of 1480-1520° during tapping.

In order to lower the viscosity of such a slag it was necessary to increase its temperature; yet this was impossible as then the silicon content in the pig iron increased. Under conditions corresponding to those in Fig. 2a, signs of unstable furnace operation appeared when slag basicity was increased to 1.30; this was explained by the attendant increase in slag viscosity which also favored an increase in the pig-iron sulfur content. When slag basicity was lowered to 1.18-1.25, the sulfur content in the pig iron went down. Consequently, production of low-sulfur cast irons of appropriate temperature from a low alumina charge is feasible when operating on relatively acid slags ($CaO:SiO_2$ = 1.18-1.25), because such slags enable the necessary heating of the crucible and are sufficiently fluid.

Pig iron which contains 3.26-3.9% silicon and which is produced on slags of 1.06-1.25 basicity has an increased sulfur content (0.035%-0.046%).

It must be mentioned that during the experimental period about 5000 tons of hot cast iron was produced on slags of 1.06-1.20 basicity. This has confirmed that production of high silicon cast irons is possible when using slags of low basicity. That this is so is in a very large measure due to the very good heating of the furnace center and crucible as the gas flow distribution is optimum.

The increase in the sulfur content in hot cast iron, when produced on slags with low basicity, may be explained as follows. When the crucible heatup is high, slag of even as high a basicity as 1.40 containing 51.0-51.5% CaO has the low viscosity of 3-4 poise at the temperature of about 1550°. Under such conditions a slag of lower basicity will also be fluid, yet the higher basicity slag will be heated through more and will bring more heat into the crucible. Accordingly, in the case of an equal silicon content in the pig iron, there is a greater heat supply to the crucible when operating on more basic slags, than when operating on less basic ones. This results in increased fluidity of not only the slag but also of the pig iron – greater fluidity in the former case. Thus, whereas the high sulfur content of the pig iron which contains 3.25% silicon, and is smelted on basic slags is the result of insufficient heat to render the slag more fluid, so at more than 3.25% silicon the increased sulfur content in the pig iron smelted on relatively acid slags results from insufficient heating of the crucible and consequently of the pig iron.

When furnaces which produce cast iron are converted to use of natural gas, whereby all operating indices are retained, the heatup of the furnace and the crucible is not decreased and the furnace operation becomes more

even and more stable. Accordingly, on an over-all basis, the pig-iron sulfur content is decreased by 15-17% because of better volatilization, better pig-iron desulfurization during a more even furnace operation and a decrease in FeO content in the lower slag. In such a case the sulfur content in the pig iron decreases with increasing slag basicity, quite independent from the silicon content in the pig iron. At a constant alumina content in the slag (5.2-7.8%) and at a slag basicity of 1.15-1.3 the pig-iron sulfur content was 0.02-0.03%, while at a slag basicity of 1.25-1.35 it was 0.01-0.02%. A stable sulfur content of 0.04% is obtained at a slag basicity of 1.10-1.18 at any silicon content. It must be noted that when operating with natural gas the over-all basicity level of slag is lowered by 5-10% compared to periods of operation without gas while simultaneously the pig-iron sulfur content is decreased.

TABLE 4

Chemical Analysis of Smelting Products

Months	Composition											
	pig iron				lower slag							
	Si		S	P	CaO	SiO_2	Al_2O_3	MnO	FeO	MgO	S	$\frac{CaO}{SiO_2}$
March	3.08	0.71	0.028	0.089	49.83	39.63	7.41	0.52	0.54	1.09	1.93	1.26
April	2.71	0.80	0.032	0.096	48.89	39.77	7.61	0.56	0.51	1.43	1.93	1.23
May	3.02	0.76	0.035	0.102	48.79	38.82	8.18	0.53	0.46	2.33	2.09	1.25
Average for the period	2.94	0.76	0.032	0.096	49.17	39.43	7.73	0.54	0.50	1.62	1.98	1.24

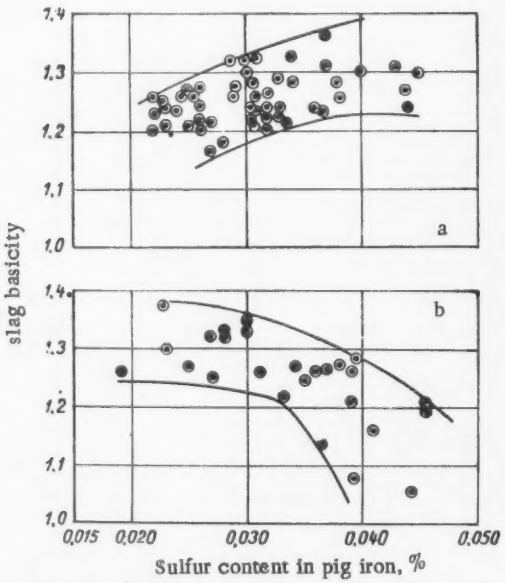

Fig. 2. Dependence of sulfur content in slag on slag basicity at the following contents of silicon in pig-iron: a) 2-3.25%; b) 3.26-3.9%.

Accordingly, it was established in the experiments on smelting cast iron with a conventional blast (without gas) and using relatively lower-basicity slags which contain 6-8% alumina, that in order to obtain pig iron containing less than 0.03% sulfur, one must operate on slags with a basicity of not more than 1.18-1.25 when the pig-iron silicon content is less than 3.25%, and a basicity of 1.26-1.35 when the silicon content is more than 3.25%. If a pig-iron sulfur content in accordance with "GOST" standards only is required, then the slag basicity may be lowered to 1.15-1.20 at any silicon content in the pig iron.

When using natural gas one must operate with a slag basicity of 1.25-1.35 in order to obtain cast iron with a sulfur content less than 0.02%. A sulfur content of 0.04% may be obtained at slag basicities of 1.10-1.18.

Starting with June 1959 all the cast iron at the Ilich Works has been produced successfully according to the above slag regime.

ON DISRUPTION IN THE OPERATION OF CRUCIBLES IN LARGE BLAST FURNACES

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In the last years there were cases of disturbances in the operation of crucibles of large volume blast furnaces (over 1000 m^3); these occurred at the metallurgical plants in the Eastern USSR. These disturbances were caused by the accumulation of refractory sediments consisting of slag and graphite which hindered the proper charge descent and gas-flow distribution in the upper furnace levels. Moreover they caused a decrease in the crucible volume and led to severe burning of the slag and air tuyeres. As a result the furnace productivity went down and performance characteristics deteriorated. In order to eliminate these difficulties and restore normal operation of the furnace much effort and time is required. Such difficulties occurred in the past, also, but they are met with more frequently now and their elimination is more difficult.

This phenomenon is, without doubt, related to the changed conditions under which present blast furnaces operate. These are the increased top pressure, the raising of the blast temperature to 900-1000°C, smelting of low-manganese pig irons, increase in the lime content of slags in accordance with the needs to produce low-sulfur pig irons.

Introduction of the above measures had led to a substantial increase in blast-furnace productivity, a high performance level (particularly with respect to coke consumption) and to the production of low-sulfur pig irons. However, such operation has brought with it a series of difficulties in the operation of the crucible.

The increased blast pressure, necessitated by the increased furnace-top pressure, has brought about a decrease in the size of the combustion zone. This should have had an unfavorable influence on the charge descent because of the reduction in the size of the oxidation zone in the direction of which the charge descends.

In addition, the increased blast pressure brings about a decrease in the molten zone and an increase in the solid zone in the crucible which leads to somewhat decreased high-temperature zones in the lower part of the blast furnace. This phenomenon is made doubly worse in the case where large blast tuyeres are used; the extent of the combustion zone in the direction of the crucible center line is diminished. This situation is confirmed by a number of examples from practice.

First Example

At one of the eastern plants, 200 mm tuyeres were installed in a 1719 m^3 inside-volume 18-tuyere furnace. When the top pressure was increased to 1.2 atm frequent cooling off and accretion of the crucible occurred. The latter were accompanied by flooding of blast tuyeres and nozzles by slag and pig iron. This resulted in the burning of tuyeres and nozzles and in lengthy furnace shutdowns. The crucible condition improved significantly only after the installation of 170-180 mm diameter tuyeres.

With the 200 mm diameter tuyeres the gas velocity and kinetic energy of the blast were 130 m/sec and 3435 kgm/sec respectively and with 170 mm diameter tuyeres these were 141.6 m/sec and 4050 kgm/sec. It is possible that some further decrease in tuyere diameter (for example to 160 mm) might give an even better temperature condition in the crucible and better over-all furnace performance.

Second Example

At the Cherepovetsk Plant, where the blast furnaces had 165 and 150 mm tuyeres and the furnace-top pressure was 1.5 atm., an irregular tuyere operation was found. Also high gas temperatures at the furnace shell were observed and the furnace was unstable in the way it held the heat. After the installation of 140 mm tuyeres furnace operation improved. According to the plant laboratory observations the optimum kinetic energy of the blast is 6500 kgm/sec.

The necessity to decrease tuyere diameter is being understood also at other plants where furnaces operate with increased top pressures. For example at the Magnitogorsk Metallurgical Combine (MMC) the tuyere diameter has been decreased from 180 mm to 165 mm on some furnaces.

Difficulties in the crucible operation arise from strong overheating of smelting products brought about by a large concentration of heat in the lower part of large furnaces, a rise in the blast pressure, use of more basic slags, and a switch to smelting low manganese pig irons. Present slags at MMC are characterized by a high basicity (the ratio $CaO:SiO_2 = 1.1\text{-}1.15$), a high MgO content (8-10%), a virtually complete absence of manganese oxide. As a result of the conditions indicated above the pig-iron temperature in the MMC-blast furnaces has increased greatly over the past years and is now 1470-1500°. It is 1440-1480° at the Chelyabinsk Metallurgical Works.

One of the reasons for the large concentration of heat in the crucibles of contemporary furnaces is the increase in the crucible diameter. Already after the blowing-in of the first large blast furnaces in the USSR (volume of 930 m^3) blast-furnace operators noticed a phenomenon which had not been seen with small furnaces, namely, tapping of pig iron containing 0.5-0.8% silicon accompanied by a strong separation of graphite. Further increases in the crucible diameter together with the above mentioned factors led to an increase in crucible temperature and extensive solution of carbon in the pig iron. Use of basic slags made a blast furnace very sensitive to changes in the heat content. The smelting temperature of such slags was high; in case of cooling of the furnace they could freeze at 1400°.

When the temperature of a strongly overheated pig iron is lowered, the carbon dissolved in it is separated out as graphite. As a result of freezing of slags and of separation of graphite in the crucible difficultly fusible slag-graphite accretions are formed. The appearance of such accretions may also be promoted by the formation in the furnace of large quantities of coke fines (if weak, easily abraded coke is used for smelting). Those suppositions are confirmed by the fact that, after blowing-out a furnace, large graphite slabs are found in the lower part of the metal collecting vessel. After blowing-out a furnace at the MMC, graphite slabs (frequently quite pure) 1100-1200 mm in thickness were discovered in the lower part of the metal collecting vessel. Removal of these accretions presents a serious problem. At the eastern plants, hearth cinder and manganese ore are used to wash them out; yet such methods do not always permit a speedy removal of these accretions.

At the MMC the large furnaces (1380 m^3) are suffering the most. This is apparently related to the use of high top pressure (1.0-1.1 atm, compared to 0.7 atm in lower-volume furnaces), use of weaker coke in the charge (Coke Oven Block I), somewhat greater forcing of the smelting process and finally to the larger crucible diameter (8 m as compared to 7.62 m). In addition it should be mentioned that the appearance of slag-graphite accretions is promoted by fluctuation in agglomerate and ore composition, changes in moisture, and blast temperature, and others.

When operating on strongly basic slags ($CaO:SiO_2 > 1.15$) increasing blast moisture or decreasing its temperature, in order to in this way influence the furnace operation (when signs of an unstable operation appear), may lead to a weakening of the crucible. Accordingly such measures should be resorted to only in cases of most urgency.

The basic source of upsets in crucible operation, related to chilling of the furnace, are disturbances in the flow of gases through the charge, i.e., formation of "channels." When channelling in the charge column is present, utilization of the chemical (and thermal) energy of the gases is decreased and the ore-portion of the charge, which prior to that time had given pig iron of desired composition, increases. The result is that the thermal balance of the furnace is upset. Unprepared materials descend into the crucible and the crucible temperature decreases. When such a course continues, difficult-to-melt graphite-slag accretions are formed, drainage of molten materials from the crucible deteriorates and slag appears in the tuyeres.

In order to avoid upsets in the crucible operation of large blast furnaces which operate with a top pressure of 1.0-1.8 atm, the following must be provided for:

(a) The possibility of channelling should be eliminated; this may be achieved by further improving the charge permeability by screening off the minus 5 mm fraction of the agglomerate, improving coke quality, and using coke of particle size greater than 40 mm.

(b) Define a more rational slag composition which would have sufficiently pronounced sulfur-removing properties and at the same time have a sufficiently low crystallization (solidification) temperature.

(c) Decrease the tuyere diameter to a value which should be determined by experiment.

(d) Exclude the possibility of temperature and moisture fluctuation in the blast when operating on strongly basic slags (it is desirable to keep the blast moisture to a minimum when operating on strongly basic slags).

IMPROVEMENTS IN THE HOT-AIR VALVE ON BLAST STOVES

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Nizhne-Tagil Metallurgical Combine
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At the Nizhne-Tagil Metallurgical Combine the joint between the valve and the connecting pipe was strengthened by increasing the bolt diameter to 30 mm and the nut thickness to 50mm, in order to eliminate leakage at the flanges (previously bolts of 22 mm diameter and 18-20 mm in thickness were used; on tightening the flanges the bolts snapped and threads were stripped in the nuts).

The round packing rope, 25 mm in diameter and made from asbestos,could not be adequately compressed when the weak bolts were used. As a result the joints were not tight and leaked. The round asbestos was replaced with a circular packing which was lap-rivetted from 80 mm wide sheet steel. Presently, the rivets are put in flush with the packing. The inside diameter of the packing is determined by the distance between the bolts on the flange bolt circle. After rivetting, the packing is wound loop-to-loop with asbestos cord (2-2.5 mm diameter). The loops of the first layer are aligned at 30° to the packing diameter; the second-layer loops are wound at 30° to the diameter in the opposite direction. The thickness of the packing is 8-10 mm. Before positioning the packing is immersed in molten glass or in fused salt and then introduced within 2-3 minutes between the flanges.

Analogous packings of corresponding size are used for sealing the tuyere apparatus, manholes, flanges, the gas-shutter, cold-blast valve, and many other locations. The strengthened bolts permit tightening the packing to 2-2.5 mm. After a day the bolts are retightened. As a result, in the period between 1950 and 1960 there were no cases of leakage between the flanges. There was no need to water cool the flanges.

Valve collars fail largely because of thermal stresses under conditions of large temperature changes. The inner collar wall has to sustain very severe conditions. When under the effect of the blast it is subjected to a high thermal stress and when the gate is closed this stress is reduced. The alternating heating and cooling of the inner wall leads to cracking of the collar corners to a much greater degree than of the rest of the wall. Depending on the quality of manufacture the rings have a life of from one to twelve months.

The system of feeding and removal of water was modified in 1954. While previously water was fed to the ring from the bottom and removed from there also, now it was begun to feed at two points from below and to remove at two points from above. The temperature drop decreased by a factor of two, and the ring life increased 2-3 times, yet the temperature load on it remained as before.

On an experimental basis, the ring of one of the valves was lined with fireclay brick decreasing its passage from 1100 to 860 mm in the form of a venturi tube. The lining stood up well and the water-temperature drop decreased to 2-3°. The valve was in service for three years and was removed because of a crack in the casing while the ring was still intact.

At the end of 1959 and beginning of 1960 the rings on three blast furnace stoves were lined with brick. The following data may serve to illustrate the effectiveness of these measures. Throughout 1959 15 valves on 13 stoves were changed while at the Dzerzhinskii Works 36 valves were changed on 18 stoves. A thermocouple installed in the connecting pipe to the valve at a point 800 mm from it showed that during the "blow" period

the temperature reaches 1120° and in the heat-up period it drops to 400-450° as a result of a gas cushion that forms in the connecting pipe. This refutes the claim that the premature failure of the ring and the gate are due to high temperature effects during the heat-up period. The suggestion that the ring fails due to the formation of vapor pockets in the upper cavity of the rings and the gate was not confirmed. The cracks on the gate are always found on the lower part while the pockets form in the upper part, where there are never any defects. The ring fails at any point along its periphery with the formation of parallel and perpendicular cracks.

The holes in the flanges on the valve and the connecting pipe should be made oval, because when they are round and happen not to fit they have to be burned out with a torch, and this fact has detrimental effects upon the valves.

The contact surface between the flange and the packing should be flat and 60-80 mm wide. Flange thickness should not be less than 45 mm as flanges of 25-30 mm bend and touch before a tight compression of the packing is achieved. In addition the flanges may break off and the outer coating of the valve casing may be broken. The apertures in the faucets in the feed line to the valve should be large in order to avoid dirt deposition. The water removed from the valve should be overflowed at a level of 1.5-2.0 m above the ring in order to ensure that there is some back pressure on the valve. It also must be insured that the possibility of draining the water from the ring does not exist when its throughput rate decreases. During each daily change the temperature should be measured and recorded for control purposes.

The gate stem is greased 2-3 times daily in order to protect the glands and prevent gas leakage. Bronze in the valve may be replaced with soft steel without degrading the reliability and the service life of the valve. Steel rings must be produced from billets on a tire mill. The number of bolts on the flange may be decreased from 48 to 38 while increasing their diameter from 30 to 32 mm. In such a case the flange strength will increase.

Use of Karpenko type[1] sight glasses necessitates shutting down the blast furnace when changing the hot-blast valves, which causes losses of blast, furnace down time and loss of cooling water. Accordingly, they cannot be recommended. Taking into account the cases of burning and fusion of stove combustion chambers because of leakage of air through loosely fitting valves, the replacement of metallic rings in the hot-blast valves with ceramic ones [2] can similarly not be recommended, because of possible leakage of hot blast into the combustion chamber.

Water cooling of connecting-pipe flanges on hot blast lines at the Makeev Works has increased their service life but has not prevented leakage at the flanges. At the Magnitogorsk Combine gate valves have been replaced by globe (disc) valves, which are cheaper and permit quick changing of parts. However, such valves do not provide a sufficiently tight fit because the alignment between disc and the seat may readily be upset, and also because of the stiff connection between the stem and the valve. In addition much dirt is collected on the disc. A very significant part of the seat and the valve are subject to the action of the strongly heated blast. Thus the disc and the seat work only for 6-8 months.

The metallurgists are in search of a more useful design of a hot-blast valve. It appears to us that the valves now installed on our stoves may be recommended to other works.

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*See English translation.

ON IMPROVING WORKING CONDITIONS IN AGGLOMERATION PLANTS

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Zaporozhstal' Works
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When the agglomeration plant at the Zaporozhstal' Works was started up in 1951, a large quantity of dust, steam and gas were generated even at a low production rate at almost all stages of the process. Most of all, there was much dusting during agglomeration. The following factors had a further harmful effect on the working conditions: the number of agglomeration machines was increased, the fluxed agglomerate began to be used (the basicity, at present, is up to 1.2), lime and pyrite cinder were added to the charge, lime-calcination machines of the "OPR" type were introduced.

At present, after the reconstruction of the equipment, the productivity of each machine has risen to 2700 tons/day. Under these conditions it would not be possible to work in the plant without taking measures to protect the working force.

At normal surrounding temperatures and in the absence of sprinklers the incoming materials are moistened at the agglomeration plant. The ore is moistened through pressurized nets with 3 mm diameter holes to a moisture of 6-8% while travelling on belt conveyors and when discharging into the bins. When the ore is charged by means of a dump car, water is brought to the discharge doors by means of a flexible hose over the charge bins.

Unslaked lime, which is metered by means of table feeders, is moistened as follows: in the bins through tubular sprinklers, at the discharge point from the table feeder by means of a high-capacity atomizing spray (up to 1000 l/hr) and in the conveyor by means of pressurized nets.

The dry coke breeze is moistened to 8% when it enters the plant by means of tubular sprinklers and pressurized nets. The moisture application is at the bin-to-conveyor transfer point before crushing in a four-roll crusher.

When lime entered the plant at a moisture content of 0.2-0.7% the dust content of the air at transfer points at the grinding stage, in hammer mills, and in screening was 100-400 mg/m^3.

It was not possible to eliminate all the dust by means of a single aspiratory ventilation system. Presently lime is moistened to 2-3% by means of tubular sprinklers and pressurized nets while going through the following equipment: receiving bins, apron feeders, conveyors, crushing bins, and hammer mills. Moistening the lime and venting have decreased the dust content of the air to an average of 17 mg/m^3.

Blast-furnace flue dust and manganese ore are brought into the plant at a moisture of 12-18% and are not moistened over and above this.

In order to prevent hang up of the moistened materials in the bins and at discharge points electrical vibrators have been installed.

In order to prevent charge particles from adhering to the return band of the conveyors and then falling off, the converyor joints are vulcanized and drum-type cleaners and double scrapers are installed at the conveyor pulleys.

Charge openings on bins are covered by means of conveyor bands (Fig. 1). Through the dump-car discharge doors air is sucked in together with the charge, and dust tends to be carried out from the bins. Accordingly, such air is removed through exhaust pipes, 600 mm in diameter, which are equipped with deflectors or through ventilating fans.

In the return-dust and blast-furnace-dust section a strong fog formed at low temperatures (+5° and below). Visibility was less than 1.0-1.5 m. The gallery structures made of metal were corroding.

At present, the fog in the return-dust and flue-dust section has been eliminated. To this end two methods were used; they are as follows:

(1) The elimination of fog in the return-dust galleries, where in addition to the fog there is also dust, is by way of an air-feed installation of high capacity (40-60 thousand m^3/hr). The latter blows in air, heated to 22-30° through ducts which distribute this air evenly. In the air feed chambers the room air is cleaned in panel-type mesh filters using oil. The spent air is removed through exhaust shafts, deflectors or axial ventilating fans. In the summer time this equipment is used to vent the galleries.

(2) In the room where the blast-furnace flue dust receiving bins and galleries are located, and where the moisture content of the material is 12-18% and thus no dust is generated at transfer points, elimination of fog is by a different method. The air in the room is held at 18° with the help of heating units. In addition, heating and air-recirculating units (radiators with axial fans) are installed in the galleries which heat the air to 22-25° and blow it down the galleries and along the direction of material travel. These units are located at a distance of 20-25 m from one another. On the gallery roofs deflectors are installed which remove spent air. This method is more economical but is applicable only in the cases where there is no dust. All agglomerating machines are covered with continous vacuum covers. Yet, even then, dust emerges frequently from under the covers in the main portion of the machine. Accordingly at present, exhaust pipes venting the main part of the machines are being installed at the plant. The air is cleaned in spray scrubbers before being exhausted to the atmosphere.

Much dust was evolved when the agglomerate was discharged into hoppers; this dust rose up and was blown by the wind all around the works and entered into all the rooms of the agglomerating plant. In 1959-1960 the first system in the USSR for the aspiration of dust from the agglomerate-unloading locations was erected at our works. In this system the air is subjected to two stages of cleaning before being exhausted into the atmosphere. The locations where the agglomerate is unloaded into hoppers are enclosed by a tight metallic shell. One system with a capacity of 60,000 m^3/hr services two of the machines. The dust-laden air (4-5 gm/m^3) is aspirated by fans through hoods and ductwork and is cleaned in a battery of twelve cyclones, (first cleaning stage) and in a scrubber (second cleaning stage).

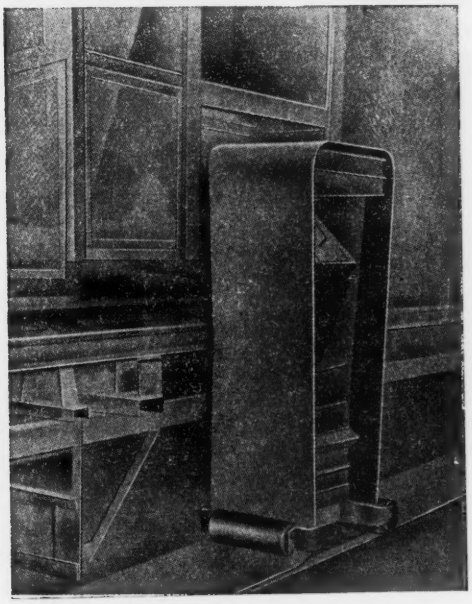

Fig. 1. Covering for the bin-charging opening.

In the cyclones 15 tons/day of dust are collected from all agglomerating machines. This dust is returned to the process. The dust content of the air which is exhausted to the atmosphere is in the normal range. After the described systems had been installed the working conditions in the discharge end of the agglomerating machines improved substantially.

When the plant was started up the dust content of the air in the multicyclone area reached 1500-2000 mg/m^3. Such a high dust content in this area, and also in the return-dust section, was explained by the fact that the method of removal of dust from the multicyclone dust bags and vertical gas ducts was not adequate. Dust was discharged through a system of two valves onto conveyors, then onto elevators which transferred it again onto conveyors. Much dust was evolved at the transfer points throughout the return-dust area. The usual methods of dust elimination, based on aspiration, did not give desired results.

In 1953 a design of a new system of dust removal from the multicyclone dust bags and vertical ducts was proposed at the works. In 1955 this method was tested in cooperation with the Mekhanobr Institute on one of the agglomerating machines.

The scheme which is presently used at the agglomeration plant is shown in Fig. 2. Water is fed to the water header which then is fed through pipes to the water traps at the bottom of the multicyclone dust bins. In the case that the outlet from the trap were to block, the excess water would be removed through the overflow pipe and to the sewer. When the operator notices the water in the gauge glass he cleans the water trap headpiece. This scheme works entirely satisfactorily.

From the water trap the water, together with the dust, flows through a funnel into the discharge pipe and then on to a collecting launder which takes the pulp from all machines into a sand-pump sump. From here the slurries are pumped into the plant slime system. For pumping the pulp two pumps of the type 2PN-4 are used (one is in operation, the other is on standby). The incline of the piping handling the pulp to the pumps is 6-10%; water consumption in each trap is 4-6 m^3/hr; the headpiece on the multicyclone dust bags is 26 mm in diameter, the vertical gas ducts are 30 mm. Such large openings have been used because, with the presence of a large quantity of lime in the charge, lumps are formed in the water trap.

The production-conditions test has shown that the system operates satisfactorily. The dust content in this section has decreased to 6-8 mg/m^3. Dust loadings have decreased significantly also in the whole return-dust area. The life of the pump casing is two months; other parts (impeller, shaft, bearings) can operate for more than a year.

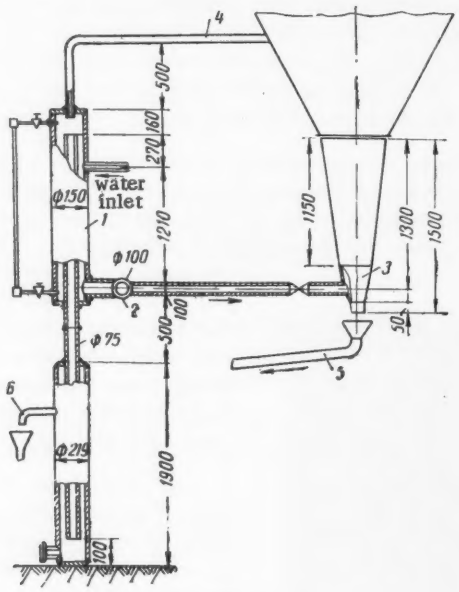

Fig. 2. Hydraulic removal of dust from the multicyclone dust bags and vertical gas ducts; 1) water header, 2) piping to the water trap, 3) water trap, 4) overflow pipe, 5) pulp removal, 6) piping to sewer.

DEOXIDATION OF STEEL WITH SILICOMANGANESE ALLOY

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During November and December 1959, 90 trial heats with the use of silicomanganese alloy from the Nova Tula Metallurgical Works for the deoxidation of steel in the furnace were carried out at the Kuznetsk Metallurgical Works.* In addition to this alloy, a small quantity (0.5-1.0 kg/ton) of ferromanganese or of 19% - ferrosilicon was added to 14 heats.

The tests were carried out in medium and large size open-hearth furnaces. The following steel grades were made: 3 sp, M16S, 3 sudost., 20, 25, 35, 40, 45, axle, St. 5, St. 6 nakl., L-53, 50, 60 kan.

To determine the feasibility and economic advantages of the use of silicomanganese alloy, the results of the test heats were compared with those of ordinary heats for the same steel grades, deoxidized beforehand with ferromanganese and 19% - ferrosilicon. The test heats and the ordinary heats were finally deoxidized in the ladle with 45% - ferrosilicon (as calculated) and aluminum (as specified).

In the test heats, to compare them with ordinary heats, one studied the effect of the method on the output of the furnace, the change in phosphorus and sulfur contents in the course of the deoxidation and tapping, the consumption of slag-forming materials (lime and bauxite), the quality of the steel, the loss of silicon and manganese, the consumption of ferroalloys and the change in the cost of deoxidation.

Characteristics of the Alloy

For the test heats grade 2 alloy and grade 3 alloy were used. According to the manufacturer's specifications their chemical composition should be, %:

Alloy grade	C	Mn	Si	P
2	3.5	40-50	8.1-10.0	0.40 max
3	3.2	45 min	10.1 min	0.35 max

Actually, the limits of the chemical composition of both alloys were, %:

C	Mn	Si	P
2.6-3.4	45.0-52.0	8.1-11.2	0.25-0.42

Compared with the ferromanganese, the alloy contained about half as much carbon, 20-27% less manganese, 6-9% more silicon and up to 0.06% more phosphorus in some batches.

As regards its physical properties, the silicomanganese alloy is very similar to the ferromanganese but it is easier to crush.

There are no slag inclusions. 100-300 mm fraction in the alloy constitutes 80%, the remainder consists of particles 20-100 mm in size.

* M. M. Privalov, V. S. Protopopov and I. F. Shinkarev took part in this work.

When one uses an alloy with a large proportion of the fraction below 100 mm, the loss of manganese and silicon increases significantly, apparently as a result of the entanglement of the alloy in the slag. Grade 2 alloy costs 1500 rubles per ton and grade 3 – 1600 rubles per ton.

Characteristics of the Process

The carbon content in the metal before the deoxidation was in general similar in the test heats and the ordinary ones.

The use of silicomanganese alloy instead of ferromanganese for the preliminary deoxidation had no effect on the steel temperature.

The temperature of the metal before the deoxidation of the test heats, as well as of the ordinary ones, fluctuated within relatively large limits, the difference between individual heats for the same steel grade varying by as much as 35° C.

The phosphorus and sulfur contents of the test heats changed during the deoxidation and tapping in the same manner as those of the ordinary heats.

The use of silicomanganese alloy for the deoxidation resulted in a reduction of the total duration of the heat because the deoxidation period was shortened by 5 minutes in medium furnaces and by 6 minutes in large furnaces; for steel grades St. 5, St. 6 and L53 the deoxidation period in large furnaces was on the average 6 minutes longer for the test heats, because in the preliminary deoxidation of steel in the ordinary heats in large furnaces, only ferromanganese was used.

The Quality of the Steel

The macrostructure and mechanical properties of the steel from the test heats were tested within the requirements of the GOST specifications.

The results of the inspection of the macrostructure of the samples taken from the end pieces of the rolled steel from the test heats and from the ordinary ones are shown in Table 1.

The quality of the macrostructure of steel from the test heats was somewhat better since there was a relatively high percentage of rejects of grade 45 steel from some ordinary heats.

The mechanical properties of all steel grades from the ordinary heats and the test ones do not differ substantially, and satisfied GOST specifications.

The Consumption of Deoxidizers

The consumption of deoxidizers was determined only for those test heats in which the deoxidation of the metal in the furnace was carried out with silicomanganese alloy. For comparison, we took those ordinary heats from the same period in which the preliminary deoxidation was made with ferromanganese and 19% ferrosilicon.

In the calculation of the consumption of deoxidizers we excluded the test heats of steel grades St. 5, St. 6 and L53 made in large furnaces because the ordinary heats were deoxidized in the furnace with ferromanganese only. An average reduction of 0.01% of manganese and silicon in the steel from the test heats was not taken into account in the calculations.

Data on the consumption of deoxidizers and on the loss of silicon and manganese in the experimental heats and the ordinary ones is given in Table 2.

The consumption of deoxidizers for the preliminary deoxidation in the experimental heats was 12.8 kg/ton, i.e., 3.6 kg/ton less than in the ordinary heats. The total consumption of deoxidizers in the experimental heats was 17.1 kg/ton, i.e., 3.3 kg/ton less than in the ordinary heats.

The manganese losses constituted 21.4% in the experimental heats, and 22.9% in the ordinary ones; the silicon losses (total) constituted 33.3% in the experimental heats and 31.4% in the ordinary ones. The small increase (1.9%) in the silicon losses in the experimental heats can be explained by an inaccurate determination by volume of the 45%-ferrosilicon, and by rather high losses of silicon in some heats because of a larger proportion of the small-particle fraction (particles below 100 mm in size) in the silicomanganese alloy.

The cost of the deoxidizers (except aluminum) in the experimental heats was 22 rubles 54 kopecks per ton of steel, i.e., 7.05 rubles more than for the ordinary heats.

Calculations show that the use of silicomanganese alloy can be introduced without economic disadvantage if its price is reduced; the price of grade 2 alloy would have to be reduced to 961 rubles and that of grade 3 alloy to 1025 rubles per ton.

TABLE 1

Results of the Inspection of the Macrostructure of Medium-Carbon Steel from the Test Heats and the Ordinary Ones

Heats	Steel	End samples inspected						
		total	passed		rejected			
					on account of central porosity		on account of impurities	
		No. of pieces	tons	%	No. of pieces	%	No. of pieces	%
Experimental	20	10	7		3		—	
Ordinary	20	8	8		—		—	
Experimental	25	5	5		—		—	
Ordinary	25	2	2		—		—	
Experimental	35	7	7		—		—	
Ordinary	35	10	10		—		—	
Experimental	40	8	7		—		1	
Ordinary	40	16	14		2		—	
Experimental	45	46	42		2		2	
Ordinary	45	62	49		12		1	
Experimental	50	6	5		1		—	
Ordinary	50	4	4		—		—	
Experimental	60 kan	2	2		—		—	
Ordinary	60 kan	2	2		—		—	
Total								
Experimental	29 heats	84	75	89.4	6	7.10	3	3.5
Ordinary	27 heats	104	89	85.70	14	13.40	1	0.9

TABLE 2

Consumption of Deoxidizers during the Melting of Steel in 190-Ton and 390-Ton Open-Hearth Furnaces

Steel	Experimental heats						Ordinary heats						
	No. of heats	consumption of deoxidizers, kg/ton			losses, %		No. of heats	consumption of deoxidizers kg/ton				losses, %	
		alloy	45%-FeSi	total	Mn	Si		FeMn	19%-FeSi	45%-FeSi	total	Mn	Si
190-ton furnaces													
St. 3sp	4	12,42	3,24	15,68	26.8	34.7	7	8,83	7,87	2.53	19,23	32.4	39,3
M16S	2	11.7	3.5	15,20	27.6	37,0	2	8,5	7,5	2,5	18,50	30.8	40.8
s/s 3 sp	4	11,6	3.5	15.10	31.6	33.4	2	8,0	8,1	2,5	18,60	32.8	43.5
20	3	11,5	4.8	16.30	18.3	35.5	4	8,2	8.3	4,6	21,10	13.7	30,2
25	6	12,5	4.24	16.74	21.9	32.3	6	7,8	8.2	3,0	19,00	18.5	30.2
35	1	13,9	4.5	18.40	27.0	41,2	1	10,3	9,9	4,2	24,40	26.4	21,0
40	1	14.3	5,10	19,40	9.8	39.5	1	10,0	7,7	5,2	22.90	13.1	34.0
45	7	14,75	4,52	19.27	20,6	31,4	5	10,0	8.3	4,8	23.10	24,6	37.8
Axle steel	9	14.2	4.82	18,40	24.6	36.3	10	10,2	8.4	4,6	23.20	20.3	27.5
St. 5sp	4	14.7	4,7	19.40	10.0	27.0	4	9,2	8.3	4,5	22,00	16.2	30.8
60kan	1	10,0	6.0	16.00	21.4	36.0	1	7.35	4.9	4,9	17.15	30.0	13.7
Total	42	mean 12.82	4,45	17,27	21.8	35,0	43	8.94	7.95	3.94	20.86	23.50	31.7
390-ton furnaces													
St. 3sp	9	12.3	2.4	14,70	21.5	29.0	11	8,76	6,12	2,67	17,55	29,0	29,1
s/s 3 sp	2	10,4	2,9	13,30	28,3	29,2	5	7,99	6,7	2,99	17,68	32.2	38.7
35	1	13.0	4.9	17,90	11.5	27.2	1	8,9	6.3	5,0	20,20	8,8	25,2
40	1	12.9	4.96	17.50	12.5	30.0	2	10,0	6,25	4,75	21,00	16.6	26.2
45	5	14,2	4.83	19.03	14,2	32.4	5	9.8	6.2	4.9	20,90	22.4	35,3
Total	18	Mean 12,56	4,00	16.56	17.60	29.5	24	9.09	6.31	4.06	19.46	21.8	30,9
Total	60	Mean 12,77	4.30	17,07	21.4	33.25	67	9,00	7,44	3.98	20.42	22.9	31.4

A NEW METHOD OF DEOXIDIZING STEEL WITH ALUMINUM

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Enakievo Metallurgical Works
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September, 1960

The generally accepted method for the deoxidation of killed steel involves a definite sequence for the addition of deoxidizers (first ferromanganese, then ferrosilicon and finally aluminum) so that they are utilized rationally and to the fullest extent, especially aluminum since if it is added early premature losses take place. If aluminum is added at the end of metal tapping, especially when small heats are tapped, particles of aluminum are always caught with the slag and substantial aluminum losses result.

At some of our works and also abroad aluminum for deoxidation is introduced by means of steel tubes. The aluminum in the steel tubes is immersed into liquid metal deoxidized with ferromanganese and ferrosilicon. By melting at a certain depth aluminum deoxidizes the metal without significant losses. However, this method makes the deoxidation process rather difficult and expensive and therefore it has not been widely accepted.

At the Bessemer shop of the Enakievo Metallurgical Works thick acid slag is not retained during the tapping of the heat. The slag gets into the ladle when the ladle is only beginning to fill with steel. A 13-13,5 ton heat is tapped rapidly and therefore all the deoxidizers should be added into the ladle as quickly as possible and at full flow of the steel so that they can be "sunk."

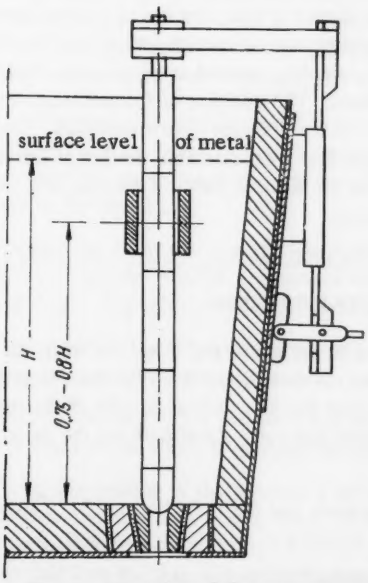

Fig. 1. Mounting of the stopper with the sleeve in the ladle.

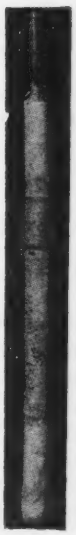

Fig. 2. Sleeve mounted on the stopper.

TABLE 1

Dependence of the Oxygen Content in Steel on the Method of Adding Aluminum

Form in which aluminum is introduced	Aluminum consumption, kg per ton of steel	Oxygen content in steel, %
Sleeves	0.5	0.008
Lumps	1.0	0.010
Ferroaluminum	1.0	0.019

TABLE 2

The Dependence of the Impact Strength of Steel on the Method of Adding Aluminum and the Temperature of Testing

Form in which aluminum was introduced	Testing temperature, °C				
	+20	0	−20	−40	−60
	Impact strength, kgm/cm²				
Sleeves (0.5 kg/t)	5.0	4.9	3.6	2.8	2.0
Aluminum lumps (1.0 kg/t)	7.2	5.7	4.4	4.3	2.3
Ferroaluminum (1.0 kg/t)	6.4	4.8	4.9	2.4	1.2

TABLE 3

The Dependence of the Impact Strength of Steel on the Position of the Aluminum Introduction and the Testing Temperature

Position of the sleeve above the bottom of the ladle, mm	Testing temperature, °C				
	+20	0	−20	−40	−60
	Impact strength, kgm/cm²				
400	3.2	2.4	1.3	1.8	1.5
1000	6.2	6.5	5.2	5.1	2.3

For the deoxidation and the alloy addition during the making of 25G2S low-alloy Bessemer steel one uses molten ferromanganese in an amount of 20-21 kg/ton, 45% ferrosilicon – 13-14 kg/ton and aluminum 1 kg/ton. All the deoxidizers are added to the ladle into the steel stream in the course of two minutes during the steel tapping, but the addition of ferromanganese and ferrosilicon is completed in 1-1.5 minutes. At the same time, in spite of this rapid rate of addition of the deoxidizers, aluminum was added at the end of the steel tapping when most of the metal surface in the ladle was covered with slag. This resulted in aluminum losses; it burnt, evaporated in the form of white smoke and sometimes was caught in the slag and even flowed through the steel-pouring nozzle after the steel pouring. To avoid these losses a new method of adding aluminum was introduced. Sleeves of aluminum weighing 6-6.5 kg were cast on a special centrifugal casting machine; they were fixed on the ladle stopper by means of a small cotter and together with the stopper were placed into the ladle (Figs. 1 and 2). During the steel tapping when the steel level in the ladle rises, the aluminum sleeve is soon covered with liquid steel. It was found that the sleeve did not rise to the surface nor did the aluminum melt prematurely.

The oxygen content in the steel was determined* when aluminum was added with the usual and the new methods, as well as when the steel was deoxidized not with aluminum but with ferroaluminum (40% Al) which was introduced after the addition of other ferroalloys. The results of the determination of oxygen (Table 1) by the method of hot extraction showed that even when the aluminum consumption was reduced by half, the use of sleeves instead of the addition of aluminum lumps into the stream of steel ensured a somewhat better deoxidization of the steel. The addition of ferroaluminum in this case did not give the results expected. Observation showed that a considerable portion of ferroaluminum rose to the surface, entered the slag and was not utilized.

The steel from the experimental heat was rolled into strip 60 × 10 mm.

Table 2 shows the results of the impact tests at room and low temperatures.

The steel from the heat which was deoxidized by aluminum in the amount of 1 kg/ton when the aluminum was introduced in the usual way had a better impact strength than the steel deoxidized by aluminum sleeves (0.5 kg/ton). The aluminum sleeves for the experimental heats were fixed at various heights of the stopper: 400 and 1000 mm above the ladle bottom. The effect of this factor was quite significant for the value of the impact strength (Table 3).

With the increase in the distance between the aluminum sleeve and the ladle bottom (i.e., a delayed introduction of aluminum into the steel) the effectiveness of the addition of aluminum increases.

* Comrades Nelepa, Kozhukhov and Malakhov took part in this work.

THE MAINTENANCE OF THE BOTTOM OF 500-TON OPEN HEARTH FURNACES

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and N. M. Savin, Senior Foreman of the Open Hearth Shop

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September, 1960

500-ton open-hearth furnaces are operated on a scrap-ore process, they are fired with coke-oven and blast-furnace gas mixture and are used for making rimmed, killed and low-alloy steel.

The output of these furnaces depends to a great extent on the service life of the furnace bottom. Therefore, special attention has to be paid to the correct operation of the furnace and proper sintering of the bottom which has to take up the impact of the charge and stand up to the action of liquid metal and steel, and the action of the flame during the fettling.

In the 4 years of operation of large capacity furnaces some experience on the maintenance on the bottom has been accumulated. The initial thickness of the sintered magnesite layer is 310 mm. If the wear of the bottom is uniform and there are no holes, the furnaces remain in operation until there is a layer of slag up to 100 mm thick on the bottom.

One should not allow the bottom to build up since then one has to employ false sills and steelmaking operation becomes difficult.

For the prevention of a build-up (incrustation) the bottom is completely covered with an even layer of ore. During the fettling of the rear wall it is important not to allow the fettling materials to fall on the furnace bottom. If the fettling of the rear wall is not finished and the fettling material (dolomite) falls off, then it is better to carry out the fetting after the ore and the layer of limestone have been charged on the bottom.

A well-sintered bottom wears uniformly; more intense wear takes place at the steel tapping hole owing to mechanical erosion by metal. Usually, one tries to keep the section of the bottom near the tapping hole slightly lower to ensure the complete outflow of metal and slag as well as the rapid removal of slag from slag pools during repairs to the bottom.

Preparation of the bottom for repairs. Before the last heat, the whole bottom is inspected through open charging doors, and the wear of the bottom is determined by dipping iron bars which measure the depth of the metal and slag. Any ridges between depressions in the bottom are blown off with compressed air.

This operation is carried out in the course of 10 minutes. The tapping hole is covered with ore so that no ridge forms in front of the tap hole and the blowing of the bottom is facilitated.

After the tapping of the last heat and the removal of slag, the tap hole is washed and one charging box of scale, free from metal scrap, is charged through each charging door in the furnace.

During the etching of the bottom, a sufficiently high temperature is maintained in the working space of the furnace so that the forming slag flows out easily. It is, however, essential that the temperature in the working space does not exceed the temperature of spalling of the chrome-magnesite roof, since otherwise brick particles may fall into the slag and make the removal of the slag difficult.

After the slag has flowed out, the slag pools are blown out with compressed air at 5 atm pressure by means of 1.25 inch tubes which are introduced into all five charging doors. The compressed air main is located on a platform opposite the third charging door.

The outer doors are opened so that it is possible to drive the slag to the tapping hole with the use of all the tubes. Liquid slag and metal is blown out of the holes in the bottom in such a way that it is not splashed on the walls and banks because any slag remaining on the walls and banks results later in an irregular wear of the bottom; when the furnace is in operation this slag melts and may cause cracks in the sintered layer.

Any metal and slag remaining on the walls is carefully blown off and then the fettling of the furnace is started.

The slag from the bottom is blown off when the furnace is not fired. If the blowing operation takes more than 30-40 minutes and the furnace cools down, the gas and air are switched on for 10 minutes to heat up the furnace and then any remaining slag which collects in the depressions is finally blown out.

If the bottom is uniformly worn the etching and blowing takes 35-50 minutes.

Fettling of the furnace. Before the magnesite powder is thrown on the bottom it is necessary to reestablish the temperature of the working space of the furnace, and for this prupose coke-oven and blast furnace gases are fired each at the rate of 5000 m^3/ hr which is equivalent to approximately 25 million kcal/ hr; at this rate of firing the furnace is heated up sufficiently in 10 minutes to permit the filling of holes. The first layer serves to fill all the depressions in the bottom. All depressions are levelled off by means of steel bars and any heaps of the thrown magnesite are blown off with compressed air.

Special attention is paid to ensuring that the magnesite layers are of uniform thickness since on this depends, to a great extent, the quality of the sintered furnace bottom. An excessively thick layer does not sinter well and may cause the formation of holes.

After a short period of heating, the second layer is used to form the necessary slope and finally to smooth out the surface of the bottom. At the same time a layer of magnesite is spread on the front wall and the banks.

After being heated for 40 minutes, the bottom is covered with a third layer up to the final required level.

The layer is heated for 30-40 minutes and then scale is spread in two uniform layers with the use of the throwing machines on the whole bottom starting from the front and rear walls of the banks. During the heating and slagging of the bottom, the rate of coke-oven gas is increased to 6,000 m^3/hr, thus increasing the total heat liberated to approximately 30 million kcal/ hr.

After the scale has melted and pools of slag have formed on the bottom, the tapping hole is opened and the slag is run off; the bottom should be absolutely even and dry and should have a slope toward the tapping hole; if there are still pools of slag, they are then covered with magnesite powder and dried and then the charging of the furnace begins without the firing rate being reduced.

When the new magnesite layer is soaked with scale, a solid working layer containing 15-20% iron oxides is formed. The mineralogical transformation of the lower layers takes place during the first heats after furnace repairs.

The three-layer sintered bottom, up to 130 mm thick, heated and saturated with scale, lasts for 26 heats, which is satisfactory for 500-ton furnaces.

Repairs to the steel tapping hole. By the time the furnace is to be repaired the tapping hole is eroded and shortened substantially owing to the wear of the rear wall near the tapping hole. In addition, during the blowing of the bottom the tapping hole is further damaged by the slag, metal and compressed air.

During the bottom repairs, the tapping hole is restored to its original dimensions. Immediately after the bottom is blown out the furnace helpers begin to break down the hole from the outside of the furnace and continue this work until the slagging of the furnace with the scale. With the introduction of the first magnesite layer the furnace bottom is smoothed out.

A wooden log of 150-200 mm diameter is placed at the necessary angle on the smooth bottom; the log is covered with dolomite on the inside of the furnace in order to prevent it from burning and to form the base

for sintering. On the other side of the wall a mixture consisting of 85% ground magnesite powder and 15% fireclay is rammed around the log.

For the sintering of the tapping hole a mixture consisting of 75% magnesite powder and 25% scale is used. This mixture is thrown into the tapping hole every 10 minutes during the sintering of the bottom layers. After the introduction of the magnesite layer, only the scale is thrown on the tapping hole. The sintering of the hole is finished before the slagging of the bottom. The hole is then cleaned of the charred log and the mixture which did not sinter, it is pierced with a bar to remove any ridges, and then the slag is run off. Sometimes before the sintering the tap hole is laid out with magnesite bricks leaving a cross-section area of 150 x 260 mm.

We quote here the average duration of individual operations of furnace repairs, hr/ min:

Blowing of metal and slag	0-40
Spreading the first layer	0-15
Heating	0-20
Spreading the second layer	0-20
Heating	0-40
Spreading the first layer	0-20
Heating	0-40
Slagging	0-30

In all, furnace repairs take 3 hours 45 minutes on the average. The idle time of furnaces during repairs was reduced from 2.7-3.2% in 1958 to 2.3-2.5% in 1959.

INTENSIFICATION OF THE STEELMAKING PROCESS IN ELECTRIC-ARC FURNACES

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"Elektrostal'" Works

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Up to now in all the various methods of making structural steel in electric arc furnaces either from fresh charge or by remelting alloy scrap, or in the duplex process, the deoxidization has usually been carried out in the same way, in the main by the application of the diffusional deoxidization of metal through the slag.

Thus for instance during the deoxidization period in the process of melting ball-bearing steel ShKh15 in basic electric-arc furnaces at the "Elektrostal'" Works the metal is deoxidized by diffusion and kept under a carbide slag for at least one hour.

40 minutes after the beginning of the finishing period, ferrochromium is added to the deoxidized metal; 10-15 minutes before the tapping the carbide slag is converted into white slag and lump ferrosilicon is added into the furnace. Before the tapping into the ladle, the metal is deoxidized with lump aluminum (0.4 kg/ton) on a rod. The total duration of the finishing period is one hour 40 minutes to two hours 10 minutes.

With the object of increasing the output of 200 ton electric-arc furnaces without impairing the quality of the metal during the production of steel ShKh15, the "Elektrostal'" Works developed and introduced a new method which differs from the old one in the following ways:

(a) a simultaneous combination of the processes of a partial dephosphorization of the metal and melting the charge by adding lime and ore into the bath at the end of the melt-down period;

(b) a lower carbon content at the beginning of the oxidation period and the removal of carbon during the boil period to at least 0.30% which ensures a satisfactory degasification of the metal;

(c) a preliminary settling deoxidation of the metal at the end of the oxidizing period with refined pig iron (containing 4.0-4.5% carbon, 8.0-10.0% manganese and a maximum of 0.030% phosphorus) in the amount of 7.5-12.5 kg/ton;

(d) an additional settling deoxidization of the metal with lump silicochromium, grade 50 (5.0-6.0 kg/ton) and aluminum on a rod (0.4 kg/ton) before making the deoxidizing slag;

(e) the addition of the main bulk of the ferrochromium to the bare metal without preliminary diffusional deoxidation;

(f) the deoxidization of the slag with the pulverized coke and 75% ferrosilicon and final deoxidization of the metal with lump aluminum (0.5 kg/ton) before tapping;

(g) the total duration of the deoxidizing period is 60-70 minutes.

Tests on nonmetallic impurities of the metal were made on 33 experimental heats in accordance with GOST 801-47 and additionally on 12 samples in accordance with the TsNIIChM scale.

Data on the Experimental and Ordinary Heats

Item	Method of steelmaking					
	from fresh charge with the use of oxygen			remelting with the use of oxygen		
	heats			heats		
	ordinary	experimental	excess (+), deficiency (-)	ordinary	experimental	excess (+), deficiency (-)
Number of heats	31	9	−22	60	16	
Duration (net) of the melt-down period, minutes	75	70	−5	102	100	−2
Duration (total) of the melt-down period, minutes	81	75	−6	116	122	+6
Electricity consumption during the whole melt-down period, kw-hr	8150	7740	−410	9920	9730	−190
Duration (net) of the boil, minutes	51	45	−6	–	–	–
Duration (total) of the boil, minutes	64	61	−3	–	–	–
Electricity consumption during the boil period, kw-hr	2470	2300	−170	–	–	–
Duration (total) or the finishing period, minutes	105	66	−39	105	66	−39
Electricity consumption during the finishing period, kw-hr	3340	2080	−1260	3100	2080	−1020
Duration (net) of the heat, minutes	222	179	−43	205	166	−39
Duration (total) of the heat, minutes	250	202	−48	221	188	−33
Idle periods between the heats, minutes	41	46	+5	43	40	−3
Electricity consumption per total heat, kw-hr	13975	12120	−1855	12930	11810	−1120
Electricity consumption per 1 ton of acceptable product, kw-hr/ton	700	625	−75	642	595	−47

Average results of the tests show that the amount of nonmetallic impurities in the experimental and ordinary heats is practically the same.

The production of steel ShKh15 in 20-ton electric-arc furnaces by the method of combined settling and diffusional deoxidization makes it possible, basically by shortening the deoxidization period, to shorten the dura-

tion of the heats from fresh charge by 48 minutes on the average, and that of the heats from remelted scrap by 33 minutes, i.e., by 15-19% (see Table). At the same time, the saving of electric power in the heats made by remelting scrap constituted 47 kw-hr/ton and in the heats from the fresh charge — 75 kw-hr/ton of acceptable ingots.

At present, this method is employed at the "Élektrostal'" Works for the production of sectional and tool steels such as: 12KhN3A, 18KhNVA, 40Kh, ÉKhVH, 60S2A and other grades.

Letter to the Editor

THE PROBLEM OF THE REMOVAL OF DUST FROM OPEN-HEARTH STACK GASES

Eng. Yu. V. Krasovitskii, NIIOGAZ

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September, 1960

One of the main factors in the intensification of the operation of open hearth furnaces is the application of oxygen. With an increased rate of firing and an increased temperature of the gas phase in the steel-melting space, the dust emission increases rapidly and reaches, for instance in 200-ton furnaces, 7,000-7,400 kg/24 hours and at the same time 20% by weight of this dust enters the flue.

The open-hearth furnace dust is distinguished by a very high dispersing power. Our investigation showed that the particle size at the inlet to the waste-heat boilers does not exceed 1.5-2 microns. As a result, a dark brown dust blanket forms above the steel works and its vicinity (the content of dust suspended in the air reaches up to 220 mg/m^3).

In the furnace itself the amount of dust increases rapidly too and it settles on the heat transfer surfaces of the checkers and waste-heat boilers.

The dust adheres to the tubes of the waste-heat boilers and reduces the steam producing capacity of the boiler. At the same time there is an increase in the resistance of the flow of gases in the furnace and in the temperature of the waste gases from the boiler; this reduces the draft in the chimney and eventually reduces the efficiency of the furnace operation.

Therefore, the cleaning of the gases upstream of the waste-heat boilers is of great importance.

However, because of the design of equipment as well as from the point of view of process conditions the existing methods of sanitary gas cleaning (electric precipitators, scrubbers, foam equipment, etc.) are not suitable for cleaning the flue gases upstream of waste-heat boilers.

Some interesting experiments on the cleaning of open-hearth stack gases at a temperature of 600-650°C with the use of slag wool filters have been carried out abroad.* However, in our view, the use of slag wool as filtering medium is not justified because the upper temperature limit for the use of these filters is 650-670°, and because of the high cost of slag wool, its high consumption, high hydraulic resistance (up to 500 kg/m^2), difficult regeneration, etc.

The NIIOGAZ** has developed a highly effective method of cleaning waste gases which are taken off the flue, upstream of the waste-heat boilers, through a special filter. As a filtering medium, it is most desirable to use quartz sand (with a particle size of 1-3 mm) with an addition of 14-16% (by weight) of ground magnesite brick of the same particle size analysis for the purpose of reducing the thermal resistance of the filtering material.

*L. Silvermann. Cleaning of Open Hearth Stack Gases. "Blast Furnace and Steel Plant" 1955, VII, 43, No. 7. pp. 735-744 and 752.

**NIIOGAZ–State Scientific Research Institute of Gas Purification for Industry and Sanitation.

The optimum parameters of the filter (filter bed: 40-45 mm thick, filtration rate referred to the total cross section of the bed: 0.5 m/sec) ensure that with average resistance of the filter of 85-100 kg/m^2 and a particle size of the open-hearth dust of not more than 2 microns, 92-93% of the dust is removed.

If the increase in the particle size in the gas is greater the cleaning process becomes more efficient.

The vertical arrangement of the filtering material is most suitable for industrial conditions. At the same time it is convenient to make up the filter in the form of parallel sections so that a fairly compact unit can be designed.

The experiments on the selection of the best method of regeneration of the filtering material indicate that the most reliable method for the removal of the dust from the filter is an intensive washing of the filter with water followed by drying.

At present, NIIOGAZ, in cooperation with the "Serp i Molot" Works and the "Giprogazoochistka" Institute is developing a practical design for this type of filter and for auxiliary equipment for cleaning open-hearth waste gases upstream of waste heat boilers. This method may also be found useful for some other industries.

Metallurg 9, 1960, p. 21

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THE ROLLING OF STRUCTURAL SECTIONS ON A 550 MM ROLL DIAMETER MILL

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The production of automobile trailers of a new type for transporting various long articles has been established at one of our factories. Sections No. 830 and 831 for these trailers are prepared from high-grade steel 25-30 according to GOST (All-Union State Standard) 1050-52. The dimensions and tolerances on the special sections No. 830 and 831 are given in Fig. 1. In working out the groove design, it was taken into account that both sections have to be delivered together in small batches, and it is therefore desirable for rolling to specify a common roll equipment for all three stands of mill 550, in this way reducing roll storage and time spent in roll changing.

From the point of view of rolling and groove design, these sections have the following disadvantages: 1) asymmetry; 2) the considerable thickness of the flanges, making it difficult to obtain a sharp angle on the top and smooth face surfaces; 3) the different configuration and thickness of the flanges on section No. 831, which even with the same coefficients of reduction causes the workpiece to twist in deviation from the rolling scheme.

Since the sections have the form of angle steel, with certain limitations the method of B. P. Bakhtinov may be used for calculation. The radii of the flanges and the height of the grooves developed were selected in the recommended limits.

Taking into account experience in rolling ordinary angle steel, the broadening for section No. 830 was distributed by increasing the projection of the short flange at the expense of the long one.

In the process of establishing the production of section No. 830, fins were formed in the separate grooves. Moreover, the angle at the top was found to be incompletely filled – it turned out to have a curvature of 4-5 mm because of the increased thickness of the flanges and the small coefficients of reduction at the place where they join. After correcting the groove design – altering the radius of the curvature on the ends of the flanges, increasing the reduction in the angle of the section – the deficiencies mentioned were eliminated and subsequently the rolling of section No. 830 went on normally.

For section No. 831, having a more inclined configuration, a developed groove design was also used, since in rolling without bending of the flanges it is impossible to alter the join and the use of tilters is undesirable because apparatus for this is not available.

It should be noted that, apart from the rolling scheme analogous to that for angle steel, there exists a scheme different in principle – joining two No. 831 sections into one symmetrical channel-type section. However, the absence of the required billet section and the marked weight loading per linear meter do not make it possible to use this variant.

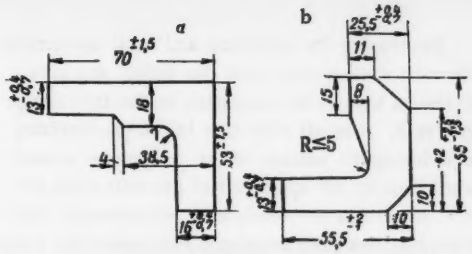

Fig. 1. Dimensions and tolerances on the special sections. (a) No. 830, (b) No. 831.

A section 120 x 120 mm was taken as the ingoing billet; box-type grooves were adopted for the first four grooves, a rectangle 73 x 85 mm being fed on edge into the fifth groove. The final pattern dimensions are shown in Fig. 2.

The presence of bevelling 13 x 14.5 mm in size on the outside surface on the end of the thick flange contributes considerably to the choice of groove design with join upward . On the upper roll there is a thin wedge-shaped crown, corresponding to this bevelling, which on feeding in the workpiece will be broken off. The grooves 8 and 10 initially used were of semi-enclosed construction (Fig. 3); these turned out to be unsuitable, since in the rolling process there was considerable overfill in the joining positions of the thick flange.

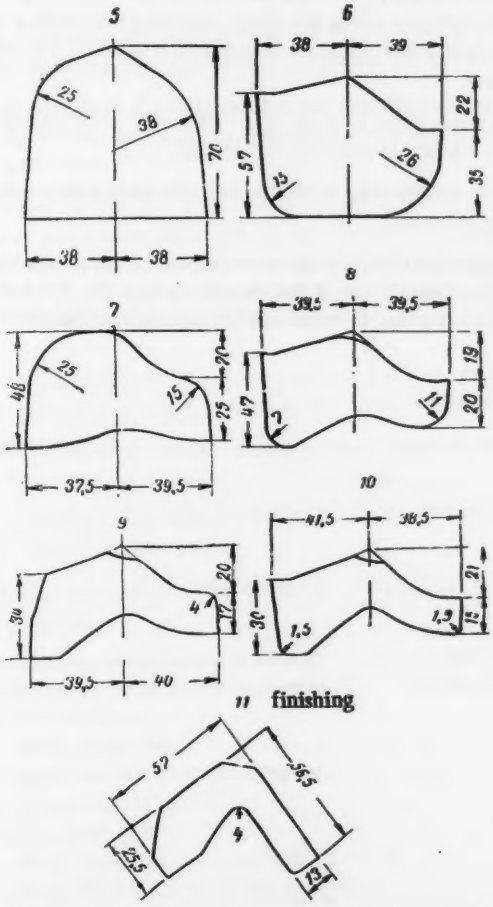

Fig. 2. Groove-pattern dimensions in rolling special section No. 831; groove numbers 5-11.

Fig. 3. Design of a semi-enclosed groove for rolling section No. 831.

Decreasing the reduction and axial movement of the rolls did not eliminate this defect and only gave rise as well to fin formation on the thin flange of groove 9. A small reduction in the pre-finishing groove during the passage of the workpiece caused forward flow by the upper roll of the mill train and severe impacts as the workpiece was ejected; this was because the chief reduction fell where the flanges join, where the rolling diameters of the upper roll are small.

Overfilling in the semi-enclosed grooves is caused by the inclined shape of the thick flange, as a result of which, even with small reductions, flow of the metal into the join takes place and this contributes to the twisting of the workpiece and makes the task of the workers difficult. Therefore, after the first set-up, a new design for grooves 8 and 10 was adopted with a top join for the thick flange without regard to its bevelling: this eliminated overfilling and waviness along the flange length in the finished section. However, some waviness of workpieces in the roughing stand could not be avoided because of the large inequality in the thickness and configuration of the flanges.

As regards the finishing stand, there twisting of the finished section was eliminated either by moving the upper roll or by regulating the flange reductions of the section.

In cooling on the cooling grid, bending of section No. 831 takes place, and it is therefore necessary to straighten it in a straightening machine.

Earlier sections of the upper and lower range were prepared from square and rectangular sections by mechanical machining, and therefore the use of the special sections No. 830 and 831 has resulted in a considerable economy with a marked decrease in metal consumption and in the cost of the automobile trailers.

USE OF THREE-ZONE REHEATING FURNACES

D. P. Galkin and A. E. Pratusevich

Sheet-rolling shop No. 1 of the Magnitogorsk Metallurgical Combine
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September, 1960

Before being rolled in the continuous sheet-rolling mill 1450 at the Magnitogorsk Metallurgical Combine, the metal is heated in three-zone continous furnaces with end charging and delivery of the metal.

The furnaces are fired with a mixture of coke-oven gas and blast-furnace gas – calorific value 2200-2250 kilocal/m^3 – and oil. The gas is fed thru six burners in the soaking zone and twelve in the heating zone. The burners are of the "tube inside tube" type.

Low sulfur oil at a temperature of 65-70° C (mainly type M-80) with a calorific value of 9600 kilocal/kg is burned in spray burners with two stage atomization (first it is atomized with superheated steam, and then with compressed air). The oil pressure is 6-10 atm, the steam pressure 6-8 atm and the air pressure 4-6 atm.

In each furnace four oil spray burners are installed (two to each heating zone) with a production capacity of 250 kg/h each.

The jets of the spray burners are directed toward the center of each row of slabs to be heated (between the pairs of skid tubes). This allows the most effective heating of the metal without marked fusion of the slab ends.

The spray burners are placed inside gas burners, which generally work on oil, although gas can also be burned in them.

The air is pre-heated in a ceramic recuperator to 300-350° C and is fed by pipes lined inside and out to the burners and spray burners of the furnace heating zones.

Brief technical characteristics and technical and economic details about the working of the reheating furnaces on mill 1450 are presented below:

Temperature to which metal is heated	1250°C
Temperature at which metal is charged	Cold metal
Metal to be heated:	
carbon steels 0-5	
motor and tractor steels 08kp-50	
electric steels É21, É3A	
Slab dimensions, mm:	
short (100-110)×(600-1000)×(1650-2050)	
long (95-130)×(630-1010)×(4500-4800)	
Width of furnace inside, mm	5452
Loading of active bottom, kg/m^2 h	
with short slabs	up to 720
with long slabs	up to 600
Temperature in the zones, °C	
soaking	1250-1270
heating	1350-1370
pre-heating	900-1100

Excess air coefficient	1.1
Recuperator heating area, m^2	830

According to the initial project, the reheating furnace production capacity was 45 tons/h. However, due to the considerable increase in the production capacity of the mill, the capacity of the three reheating furnaces turned out to be inadequate. Moreover, in the process of operating the furnaces a number of constructional defects was revealed. To eliminate them and to increase furnace production capacity, the following measures were taken.

1. Instead of a roof hung sectionally from special beams of heat-resistant steel, a roof design was used with each brick individually suspended. This made it possible to increase furnace roof life, to simplify roof construction and to reduce the duration of repairs.

2. The construction of the inclined skids at the delivery end of the furnace was changed, and this improved their watercooling.

3. To prevent cases of slabs jamming when sliding out of the furnace, on the ends of the inclined plates (with surface wear on the inclined skids) two additional inclined skids were installed between each basic pair.

4. The construction of the horizontal hearth skids of the soaking zone bottom was changed.

5. Instead of the twelve oil spray burners that had been planned for the heating zone, with a production capacity of 125 kg/h each, four spray burners with two-stage atomization and a production capacity of 250 kg/h were installed. The soaking zone and eight burners in the heating zone were changed over to gas heating. All this, besides increasing furnace production capacity, ensured a reduction in the specific fuel consumption from 130 to 80 kg of comparision fuel per ton of hot-rolled product (figures for 1951 and 1959 respectively) and reduced the fusion of metal during heating.

6. The amount of air fed into the furnace for the combustion of the fuel was increased from 30-33 to 40-42 thousand m^3/h as a result of the reconstruction of the exhaust fan driving gear (increasing the speed of rotation of the rotor).

7. The use of slabs of two lengths in the mills assortment improved the use of the active hearth area of the furnace.

8. Exhaust slide-valves, installed directly after the recuperators, were taken to smoke stacks with the aim of improving working conditions and increasing their service life (the temperature of the exhaust gases at the place where the valve is installed was lowered).

As a result of the measures taken the production capacity of the three reheating furnaces was increased to 200 tons per hour. The heating schedule with normal working is shown in the table.

However, the increase in the heat loading in the furnace to 35 million kilocal/h lead to fusion of the roof of the heating and of the pre-heating zones, and also to the rapid choking of the recuperator gas channels.

Fusion of the roof. Because of the fusion of the roof a large number of nonmetallic inclusions appeared in the flat rolled surface, and this had a bad effect on the quality of the finished product, especially on metal intended for subsequent cold-rolling.

Metal Heating Schedule

Mill working rate	Fuel consumption in the heating zones							
	soaking zone		upper heating zone			lower heating zone		
	gas, m^3/h at NTP	air, m^3/h at NTP	oil, kg/h	gas, m^3/h at NTP	air, m^3/h at NTP	oil	gas, m^3/h at NTP	air, m^3/h at NTP
Rapid	2200—2500	5500—6000	1000—1500	3500—4000	15000—18000	500—800	3500—4000	14000—15500
Medium	2200—2500	5500—6000	1000—1200	3000—3500	14000—16500	400—600	3000—3500	13000—14000
Slow	1700—1800	4200—4500	600—800	2500—3000	13000—15000	300—400	2500—3000	12000—13000

Investigations carried out in 1959 showed that the main reason for roof fusion was the breakdown of the crystal lattice of fireclay (mullite) by oxides of the alkali metals (sodium, magnesium and potassium), contained in the oil in large quantities.

The roof of the soaking zone, heated only by gas, did not show any traces of fusion.

To reduce furnace roof fusion and to lower the number of sheets with roof brick rolled into their surface, the following measures were carried out on the mill:

1. Oil spray-burners of production capacity 500 kg/h were replaced by ones of lower production capacity-250 kg/h.

2. Low sulfur oil (less than 2.5% S) was used.

3. The amount of gas burned in the heating zone was increased.

4. The angle of inclination of the spray burner jets to the horizontal was increased.

5. Water-jet furnace scale breakers were used before the roughing scale breaker.

6. A new roof material – magnesite-chrome brick, prepared from waste products – was tried out.

Magnesite-chrome bricks were installed in the inclined part of the heating zone of one of the furnaces, opposite to the jets of the oil spray burners, and in half-a-year's use in these positions no traces of fusion or breakdown were noticed.

All these complex measures allow somewhat greater stability of the roof and improve sheet quality.

Experience in the use of reheating furnaces makes it possible to draw a conclusion about the expediency of using a fireclay brick heating zone roof when heating the furnace with oil. It is probable that to reduce roof surface fusion and to improve the quality of the metal being rolled it is necessary to use other more heat-resistant materials (of the magnesite-chrome type), which can withstand a large number of heat changes.

Use of the recuperator. In heating the furnace with gas the life of ceramic recuperators is very long and stands at not less than ten years. In firing the furnace with oil, especially containing sulfur, the tubes of the recuperator are severely choked with a deposit of sulfur and by the products of chemical reaction between alkali in the oil and the tube material.

The character of the breakdwon of the ceramic tubes and their choking is shown in the figure.

With high production working of the furnace the recuperator tubes are slagged up and disintegrated in 2-3 months. The recuperator drag increases from 10 to 30 mm of water, the pressure in the working volume of the soaking zone is increased to 4-5 mm of water and furnace maintenance becomes impossible because flames escape from doors and the temperature of the surroundings rises to 80°C. The furnace draft becomes poor, and this leads to a marked lowering of its production capacity and to premature shutdown for recuperator pipe cleaning.

The characteristics of the change in recuperator drag make it possible to observe that the slagging-up and breakdown of new recuperator tubes takes place more severely than with a recuperator that has been a long time in service. With increase in the heat loading of a furnace, heated by oil, the severity of the slagging-up and breakdown of recuperator tubes increases.

To increase the service life of the recuperator between the cleaning of its exhaust tubes, exhaust gas outlets from the slag pocket of the pre-recuperator volume were made directly into the exhaust flue. Each furnace has two gas mains, calculated for drawing off 25-30% of the combustion products. The temperature of the combustion products at the base of the smoke stack has been raised very slightly and has reached 650-700° C.

These gas mains, covered with fireclay plates after recuperator cleaning, are opened without shutting down the furnace when the pressure in the working volume increases above 3 mm of water. With the exhaust outlet bypassing the recuperator, the service life between cleanings was successfully prolonged to 3.5 months.

Of course, the outlet of part of the combustion products bypassing the recuperator leads to some increase in the specific fuel consumption when the furnace is working with the bypass gas-mains open. But the economic desirability of using them is verified by the increased length of furnace working campaigns.

Recuperator cleaning in the summer time (from the moment the furnace is shut down to the start of delivering heated metal) takes three days. The tubes are first of all cleaned with compressed air (pressure up to 6 atm), and the walls are cleared of deposit and sintered masses with metallic sockets suspended on cables and lowered into each tube. The inside surface of the cleaned tubes are inspected with the help of electric lamps.

The cleaning of the recuperator occupies 8-12 hours and is combined with the daily repair of the mill equipment or with a systematic furnace repair.

During the cleaning of the recuperator, furnace scale is removed from the slag pocket and flue. All these operations are very laborious and are carried out in short periods in hot conditions.

Experience in using ceramic recuperators on reheating furnaces heated with oil indicates that the use of a ceramic recuperator with gas tubes vertically situated is not desirable; other designs of recuperator should be used, making it possible to clean the gas tubes without shutting down the furnace. It is necessary to consider the problems of having reserve recuperators by taking an additional recuperator into a separate flue. Moreover, a larger reserve of furnace draft should be provided for.

The repair of reheating furnaces. The production capacity and successful working of the rolling mill largely depend on the condition of the reheating installations. Therefore great attention is paid to the planning of furnace repairs on mill 1450.

Each furnace is repaired twice a year, which is made necessary by the service life of the horizontal skids of the soaking zone floor.

The duration of a planned repair is determined by its volume and stands at 56-64 hours, including the cooling down and reheating of the furnace. The duration of a furnace repair with full replacement of the ceramic recuperator does not exceed 96 hours.

Before shutting down the furnace for repair, a special inspection of the furnace elements is carried out with the object of determining its technical condition, and repair catalogs and an operational graph of the execution of the repair are compiled, which are discussed in a conference on the furnace repair.

The chief condition for shortening the duration of furnace repair is, perhaps, rapid cooling. For this, directly after cutting off the supply of fuel and inserting a plug in the gas main, the following tasks are carried out:

a) Slabs are removed from the soaking zone hearth with a special mechanism;

b) If it is necessary to replace the hearth skids, the roof of the soaking zone is knocked in;

c) The horizontal part of the roof of the heating zone is knocked in;

d) One or two breaches are made in the walls of the upper heating zone and ventilating air is introduced through them;

e) Water is abundantly poured on the furnace slag pocket, on the slabs situated on the skid tubes, on the hearth of the soaking and heating zones, and also on the brickwork of eroded lining;

f) All covers and inspection windows of the furnace are uncovered;

g) Slabs and skid tubes are rapidly removed.

In our opinion, to shorten the furnace repair time it is practicable to destroy even serviceable parts of the furnace roof and wall, since this creates the necessary conditions for rapid furnace cooling.

In sorting the demolished brickwork, unbroken normal bricks and the metal construction of the suspended roof should be sorted out and used again in the lining.

Treatment of the soaking zone hearth. The absence of obligatory treatment of the hearth and the infringement of technical instructions about heating the metal lead to the formation of products of the physicochemical reaction of fusion of metal, hearth scale and material of the hearth. The hearth becomes uneven, the conditions for movement of the metal are impaired, tilting of the slabs on the hearth takes place, they jam during withdrawal from the furnace and, as a consequence, there are losses in mill production capacity because of the shutdown of the furnace while the hearth is cleared, which, with reheating of the furnace, occupies eight hours.

To maintain the hearth in good conditions, the technical instructions about heating the metal should be strictly observed and it should be cleaned every day with a so-called figured slab of channel section made of cast steel. The area of the hearth is cleared of irregularities in 30 minutes without substantially lowering the mill's production capacity.

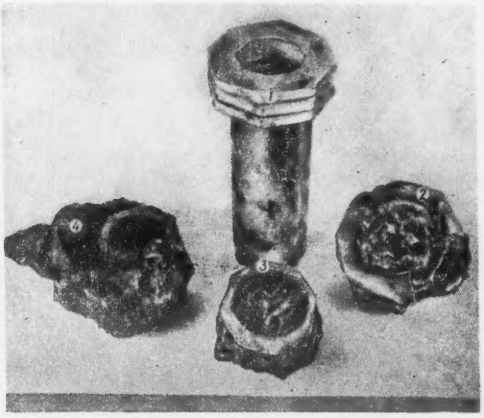
At the moment, experience is being gained on cleaning the hearth with a pneumatic hammer. The tool for clearing the hearth is water cooled and is made from tube drill steel.

The installation of two additional inclined skids between the main skids cut out instances of slabs jamming while being pushed onto the ends of the inclined plate in being removed from the furnace. The installation of horizontal hearth skids 100 mm above the refractory lining of the bottom and 30-50 mm above the level of the inclined skids increases the life of the hearth skids and reduces the number of slabs jamming during their removal from the furnace.

Covering the furnace bottom with magnesite powder has given good results, as has also the use of heat resistant steel Kh25S3N for the preparation of the hearth skids.

The measures listed above ensure that the hearth is in a reliable condition during 10-12 days working, at the end of which the mill repair is planned and the hearths of all three furnaces are cleared. Each furnace is equipped with an electric hoist, which pulls out slabs from the soaking zone to the under-furnace roller track side.

During the use of the hoist, one of the most laborious and dangerous tasks connected with cleaning the hearth occurs, namely, tilting the slabs with the help of tilting "shovels."



Type of breakdown and choking of ceramic recuperator gas tubes. 1) Tube before use; 2) formation of a deposit in the tube (carbon-fireclay and oil constituents); 3) formation of plugs from carbon, oil and disintegration of the tube walls; 4) formation of carbon deposits on outside of tube.

Organizations concerned with the planning of reheating furnaces must solve the problem of mechanizing repair work on the furnaces (extraction of slabs from the furnace, the supply of materials, the removal of broken bricks, slag and scale), and also the use of special refractory blocks for the construction of certain furnace elements. All this will make it possible to raise the development and safety technique in carrying out repair work, to speed up the execution of the repair and in the long run to increase the production capacity of the mill.

Moreover, the problems of mechanizing hearth cleaning in the soaking and heating zones and selecting a refractory material for hearths which will not react with scale and molten metal must be solved.

All these questions are very important for ensuring that reheating furnaces work reliably.

IMPROVEMENT OF AUTOMATIC LUBRICATION PUMPS

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Makeevka Metallurgical Factory
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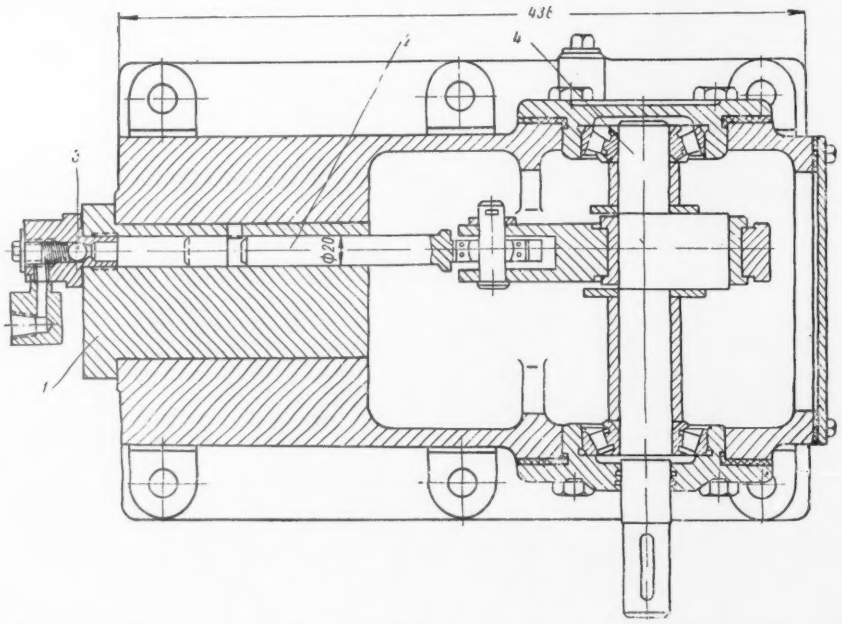
September, 1960

In No. 1 rolling shop in our factory an automatic lubrication installation SAG-500 is in operation. The pump in this installation takes the form of a cylinder with two plungers, an intake and delivery plunger. The replacement of the cylinder and plungers and the regulating of the plunger group presented difficulties and required increased skill of the maintenance personnel.

Following the suggestion of the designers V. V. Gavrilyuk, E. V. Darovskikh and G. T. Zhabotinskii a new plunger-group design was developed (see Figure).

In the cylinder of the new group only one channel for one plunger was drilled, and a ball-type nonreturn valve was installed before the lubricant outlet into the delivery manifold.



Pump with single-plunger cylinder. 1) Cylinder; 2) plunger; 3) ball-valve; 4) eccentric shaft.

The redesigned pump has been in operation in the shop for more than a year. The pressure of lubricant in the manifold is 90-100 kg/cm^2. The single-plunger pump has the following advantages over the double plunger pump:

1) the cost of cylinder group preparation is halved;

2) the necessity for a second connecting rod and ball-race No. 1303 no longer arises;

3) plunger-group replacement is speeded up and it is no longer necessary to regulate it .

ESTABLISHING TUBE PRODUCTION IN A 120-TON DRAWING MILL

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Pervoural'sk New Tube Factory
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September, 1960

The 120-ton draw mill (Fig. 1) of the Pervoural'sk New Tube Factory, constructed by the Irkutsk Heavy Machine Factory has the following characteristics:

Draw length, tons	120
Billet strength, m	4-12
Available draw length m	15
Diameter of drawn tubes, m	100-250
Drawing speed, m/ min	4-12
Return speed of plyer, m/ min	15-45

The mill has a number of typical features which distinguish it from earlier well-known designs (Fig. 2). Thus, for mandrel-drawing it is equipped with two liftable channels, which are set alternately in the drawing axis by two pneumatic cylinders.

The force from the cylinders is transferred through the toothed sector and cog-wheels to the shaft to which are secured equal-arm cranks supporting the channels on a cantilever.

In each channel there is located a rest with a clamped rod, on the front end of which is a mandrel-plug.

The tube from the table (Fig. 3) proceeds to the rollers, from where by means of the charging mechanism (plyers with rest, moving along a monorail by a rope drive) it is transferred to the rod and mandrel.

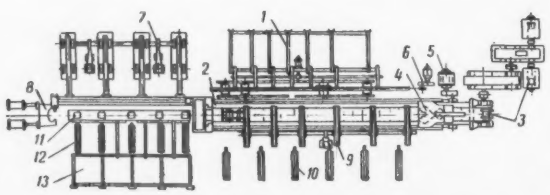

Fig. 1. Diagram of a 120-ton draw mill. 1) Charging rack with measuring hopper; 2) working table with front stand; 3) main drive and draw chain; 4) plyer; 5) plyer return mechanism; 6) mechanism for feeding billets onto rod and mandrel; 7) liftable arms for moving tubes onto the draw line; 8) mechanism for advancing tubes to the draw plate; 9) dischargers for drawn tubes; 10) container; 11) lifting channel with rollers; 12) containers; 13) charging rack.

Then the channel and tube are set up on the drawing axis, the rod and mandrel with the tube secured to them are pushed into the drawing die by a special hydraulic cylinder and the process of drawing begins; this does not differ from the usual process. At this time the next tube (billet) proceeds from the charging table onto the rollers and is moved along them by the charging mechanism onto a rod and mandrel, these being situated in the second channel.

At the end of drawing in the first channel, the second engages in its place and the whole cycle is repeated.

The mill is equipped with two plyers: one is intended for a drawing force of 120 tons, and the other for 30 tons. Depending on the type of work carried out, one of the plyers is set up. With the 120-ton plyer the drawing of long-dimension tubes

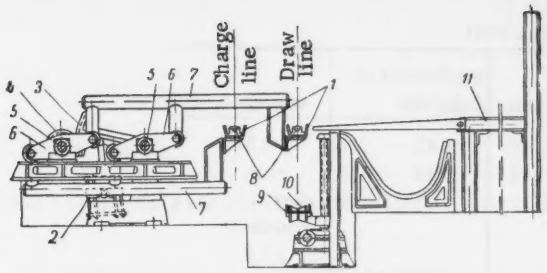

Fig. 2. Entry side of mill. 1) Liftable channel; 2) pneumatic cylinders; 3) toothed sector; 4) cog-wheels; 5) shaft; 6) cranks; 7) cantilever; 8) channels; 9) liftable channel; 10) channel rollers; 11) table.

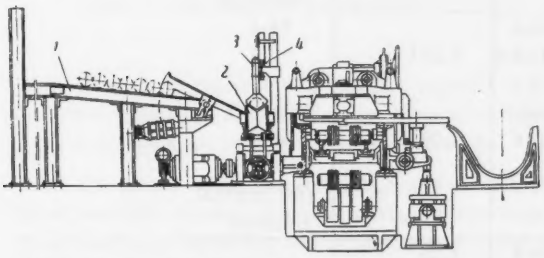

Fig. 3. Exit side of the mill. 1) Table; 2) rollers; 3) plyer; 4) monorail.

on a short mandrel is chiefly carried out: with the 30-ton plyer, drawing without a mandrel, profiling and expansion are performed. According to the authors' design for the construction of the 30-ton plyer, the head of the tube from the moment of applying the drawing force should be adequately held by the plyer pincers.

In use it was clear that it was necessary to clamp the head of the tube and to move the plyer and tube considerably before the formation of the drawing force. For this a pneumatic cylinder was installed on the plyer.

The charging mechanism and the dischargers for the drawn tube were also reconstructed, after which they began to work more evenly and reliably.

In drawing without a mandrel, in profiling and expanding, the movable channels are situated one above the other, that is, in the rest position. On the drawing axis, a liftable channel is installed (Fig. 2), with notches in which are grooved rollers; into this the billet from the table is fed, the table being situated to the right (according to the direction of drawing) of the liftable channel.

The main drive to the mill consists of two type MP82 motors of 100 kw each, working on a single shaft at 0-570 revolutions per minute.

At the moment high precision tubes are produced on the 120-ton drawing chain, prepared with internal diameters of several sizes: 108×8 mm, 145×8 mm and also a special thin walled tube 146×1.5 mm. The tubes are prepared according to the drawing specifications set out in the table.

The preliminary operations (pointing the leading ends, pickling, phosphating, annealing, straightening, etc.) are carried out in the same way as for the drawing of other tubes.

More accurately prepared tubes of the dimensions mentioned have to be obtained by new methods, and therefore the setting up of the 120-ton chain equipment made it possible to fill orders for tubes required in industry.

Some figures about the preparation of these tubes (the expenditure coefficient is given only for processing the hot-rolled tube to the finished cold-drawn product) are given below:

Tube size, mm	108×8	145×8	146×1.5
Yield of finished product, %	91.5	95.5	96
Expenditure coefficient	1.22	1.28	3.24

The figures set out show that obtaining high accuracy tubes requiring the introduction of the additional operation of drawing on the 120-ton chain and a certain increase in metal consumption necessitated by the operations preliminary to drawing (pointing the tubes ends which are subsequently cut off) is fully possible and expedient.

High-Accuracy Tube Production on 120-Ton Draw Mill

tube dimensions, mm	steel	billet, mm	reductions	coefficient of reduction	tolerance, mm
108x8	20	121x12	I–112x9	1.41	+0.02 –0.0 (for mandrel) +0.0 –0.1 (for wall thickness) +0.23 –0.0 (for internal diameter)
			II–108x8	1.15	
145x8	40Kh	159x9 (anneal)	Anneal I–151x8.4	1.127	+0.1 –0.0 (for mandrel) +0.53 –0.0 (for internal diameter
			Anneal II–145x8	1.09	
146x1.5	10	168x5	I–163x4	1.28	
			II–159x3.2	1.275	
			Heat treatment		
			III–155x2.5	1.31	
			IV–150x1.9	1.35	
			Heat treatment		
			V–146x1.5	1.21	

WAYS OF IMPROVING BLOOMING AND SLABBING MILLS

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Translated from Metallurg, 1960, No. 9, pp. 29-32,
September, 1960

In Metallurg, No. 12, 1959, an article by L. V. Andreyuk "On a rational scheme for the layout of a blooming mill" was published as a matter for discussion. A reply to that article is given below.

The correct choice of the type and capacity of blooming and slabbing mills is very important since the successful working of an entire factory depends on them. Therefore the discussion of this question started by L. V. Andreyuk's article is of considerable interest.

The growth of the production capacity of mills producing the finished rolled products outstrips the capacity of two-high reversing blooming mills, and this leads to the necessity for increasing their production capacity by the method of installing additional stands. On No. 2 blooming mill of the Magnitogorsk Combine the growth in the production capacity of the finished-product mills required the installation of two, and then two more stands; two additional stands have been installed on the blooming mill of the Makeevka factory; an additional two-high reversing stand has been installed on No. 3 blooming mill at MMK; the installation of a second two-high reversing blooming mill is being carried out at KMK and so on.

However, the installation of additional stands can eliminate the disparity between the production of the semi-finished and the finished product only for a certain period, since an adequate reserve capacity is not ensured. The continuous blooming mill* has these possibilities to an adequate extent. It can be designed for rolling any ingots to blooms of various sections. Such blooming mills have been worked out by us for rolling blooms of various sections from ingots weighing 5.0, 7.5, 9.0 and 10.5 tons.

An ingot weight of 7.5 tons had been adopted by us earlier. It should be fixed more exactly depending on the convertor or open-hearth re-allotment adopted in the factory: if necessary the ingot weight could be increased to 10 tons.

In rolling killed or unkilled steel ingots weighing 7.5 tons to blooms 250x250 mm, the continuous blooming mill will consist of eight two-high stands, in each of which a single pass is made.

For the first continuous blooming mills ordinary rectangular grooves should be adopted for all stands. Two grooves should be situated on the rolls of the first stand, and on subsequent stands, three to four each. This will secure the necessary working period between roll changing. The dimensions of the first groove should be selected from the rolling conditions of the largest-section ingot. The reduction of smaller section ingots can be started in subsequent stands, or two grooves can be situated on the rolls for rolling larger or smaller ingots (Fig. 1).

A diagram of a continuous blooming mill is shown in Fig. 2. All the stands of the mill are two-high nonreversing, located one after another at an interval to secure independent rolling in each stand: this is more convenient than simultaneous rolling in two stands (taking into account the large cross section of the product being

* A. A. Aleksandrov. Multi-stand blooming mills. Transactions of a Scientific and Technical Conference on the working of blooming mills in Soviet factories, held in Dnepropetrovsk 7-10 December 1954, Dnepropetrovsk 1956.

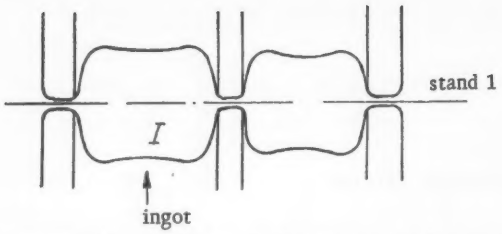

Fig. 1. An alternative layout for the rectangular grooves in the rolls of stand 1.

rolled). Therefore, in the detailed development of the mill the feasability of placing the final stands in pairs should be checked (depending on the bloom being rolled), as was suggested by L. V. Andreyuk. The rolls of the first four stands have a pitch of 400 mm, and the final ones, 250 mm.

Tilting is carried out after stand I to take off the taper from the ingot, and then after stands III, V and VII in which the product is rolled to a rectangular section. After stand VIII blooms 250x250 mm are obtained; after VII 250x350 mm (which is equivalent to 295x295 mm); after VI, 350x350 mm and so on. Rolling is carried out in all the grooves without free lateral spreading.

The average coefficient of reduction is 1.28, and the smallest, in the first two stands 1.17: in the last two the reduction reaches 100 mm in each pass (the reduction can be further increased).

The rolling speed adopted varies from 0.16 m/sec in the first stand to 1.0 m/sec in the last. The blooming mill can work in conjunction with other section rolling mills. In Fig. 3 is shown a diagram of the layout of a continuous blooming mill and continuous billet mills 700-480 and 700 for serving section mills (including wire mills). The need for two continuous billet mills is determined by the rolling speeds in the blooming mill and in the billet mills, as well as by the flow of materials in the shop. The layout of the cooling grids in the storehouse is determined by the composition of the final production mills.

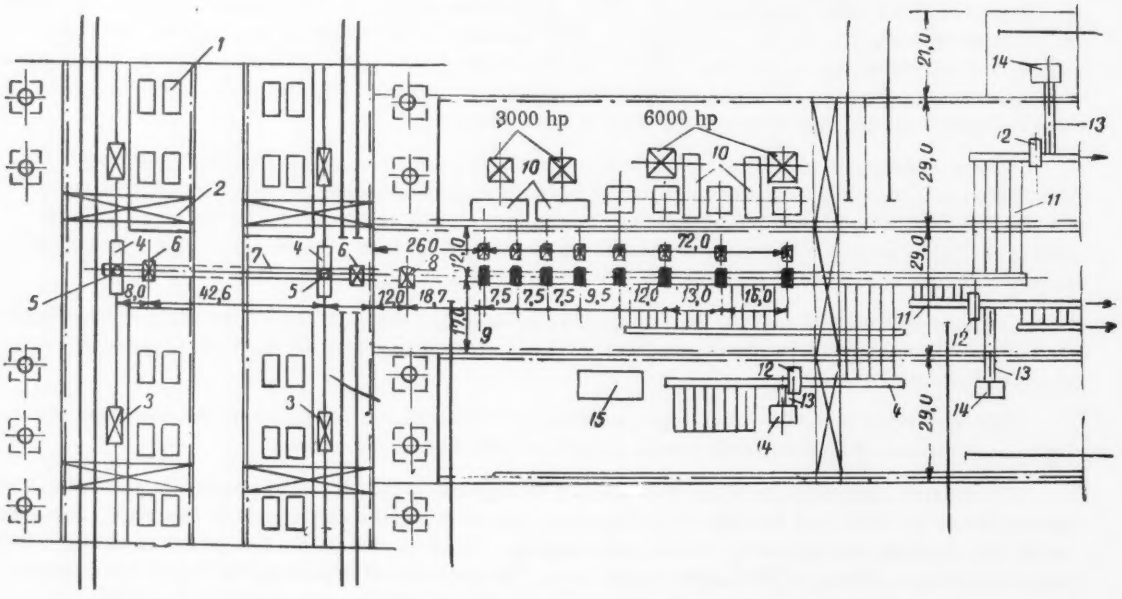

Fig. 2. Diagram of the layout of a continuous blooming mill for rolling ingots weighing 7.5 tons to blooms 250x250 mm (A. A. Aleksandrov, author's certificate No. 94063). 1) Soaking pits; 2) ground-type cranes for lid removal; 3) bogie ingot-chairs; 4) receiving roller table; 5) tilting installation; 6) stationary ingot-chairs; 7) feeding roller table; 8) ingot scales; 9) eight-stand mill; 10) reduction gears; 11) racks with dragging devices; 12) shears; 13 and 14) conveyers and bins for scrap cuttings; 15) bin for mill scale.

Fundamental Comparative Data for Different Types of Blooming Mills

Detail \ Type of mill	two-high reversing					continuous	
	single stand (1)	with stands A and B (2)	with stands A, B, C and D (3)	with six stands (4)	twin (5)	at full capacity (6)	at 65% load (6a)
Ingot weight, tons	7.0	7.0	7.0	7.0–8.0	7.0	7.50	
Bloom cross section in the stand determining production capacity, mm	240×240	280×280	345×315	420×260	350×350	250×250	
Practical production capacity of mill, millions of tons per year	2.0	2.7	3.2	4.2	4.5	7.0	4.5
Number of groups in soaking pits	8	10	12	16	17	26	17
Tentative capital expenditure in constructing entire shop, %	100.0	114.0	133.0	167.0	181.0	214.0	173.0
Expenditure per unit of final production (ratio of row 5 to row 3), %	100.0	85.0	83.5	79.3	80.3	61.5	76.2
Productivity of labor, %	100.0	132.0	150.0	167.0	166.0	236.0	166.0
Weight of mechanical equipment of blooming mill, tons (approximately)	6500	7000	8000	9500	11000	10000	10000
Production capacity of blooming mill per ton of mechanical equipment: tons per ton per year	308	385.0	400.0	442.0	410	700	450
%	100	125.0	130.0	143.5	133	228	146

Continuous blooming mills, by ensuring the necessary variety of products, have great advantages over two-high reversing mills, making it possible to carry out rolling without free spread, with large reduction and with only the bottom end of the ingot in front: this ensures that the quality of the metal is better and reduces the coefficient of expenditure. Moreover, it will have simple mechanical and electrical equipment, it will require a building with smaller bays and cranes of smaller lifting capacity; automation and mechanization on the mill will be accomplished with infinitely smaller expenditure and much more reliably. All this considerably reduces the cost of building a continuous blooming mill (L. V. Andreyuk also agrees with these considerations).

As can be seen from the table, the continuous eight-stand blooming mill will have a projected production capacity of about seven million tons, compared with the production capacity of existing blooming mills of up to 4.5 million tons.

At the new blooming mills, soaking pits of larger production capacity will be built: this will reduce their number and somewhat simplify the feeding of ingots to the mill.

From the table it can be seen that in production capacity (line 3), capital expenditure per ton of final production capacity (line 6), labor productivity (line 7) and production capacity per ton of mechanical equipment installed (line 9), the best figures are given by the continuous blooming mill (variant 6), even when used at 65% capacity (variant 6a).

In this way, modern section-rolling shops should be rated at a production capacity of not less than four million ingot tons, a continuous blooming mill being installed in them. The effect of using them will be greater, the higher the loading – column six, seven million tons per year.

An interesting scheme for a continuous mill, shown in L. V. Andreyuk's article, is calculated on the possibility of rolling blooms and slabs without changing the rolls. Such a mill can be used in a factory with a mixed variety of products. But in the scheme put forward by L. V. Andreyuk, it is impossible to roll slabs wider

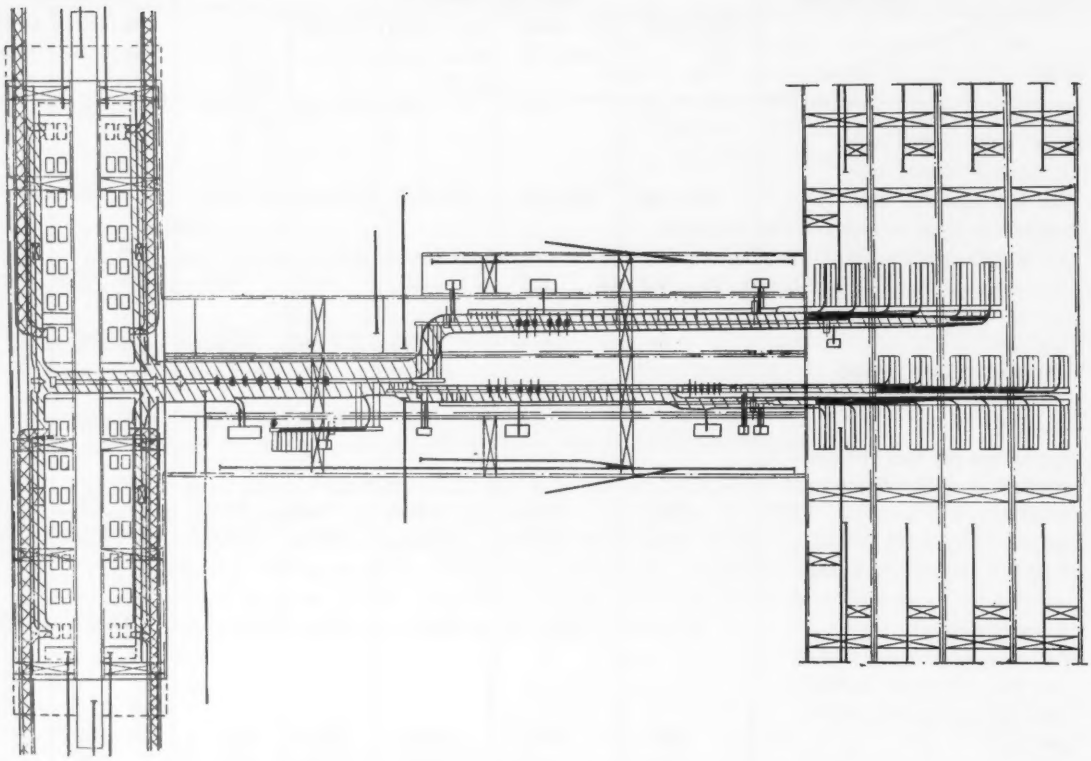

Fig. 3. Diagram of flow of material in a rolling shop consisting of a continuous blooming mill and two continuous billet mills 700-480 and 700.

than 1000 mm, which appears to be its essential deficiency: also the desire to combine in one mill the rolling of blooms and slabs has made the scheme more complicated and the mill more expensive.

For a future factory with a mixed variety of products (sections and flat) it is more practicable to use the universal two-high reversing blooming-slabbing mill (roll diameters 1250-1300 mm), which is being realized in the Soviet Union.* Broader slabs (up to 2000 mm) can be rolled on this mill from ingots of large weight (up to 40 tons) for modern flat-rolling shops with plate and widestrip mills.

This does not exhaust the possibilities of universal slabbing mills. Increasing the diameter of the horizontal rolls further and raising the capacity of the motors will make it possible to roll wider slabs with greater reductions and with a greater production capacity; there is the possibility of rolling two slabs simultaneously; the use of two-stand universal slabbing mills is not excluded.

For high production capacity rolling shops, the complete answer to the problem of guaranteeing a high capacity in the cogging resources of metallurgical factories is the construction of continuous blooming mills. In developing them, the schemes shown in Figs. 2 and 3 may be recommended.

The use of continuous blooming mills of smaller capacity for rolling ingots weighing up to 3.5 tons is also practicable for new special steel factories. Such a blooming mill, constructed similarly to that described above, having clamping equipment on all stands, will make it possible to roll ingots of various special steels with an adequately broad schedule of reductions. The essential advantage of such a blooming mill consists in the possibility of rolling in it either without free lateral spread in rectangular grooves or according to the "oval–

* A. I. Tselikov, E. S. Rokotyan, É. R. Shor. New technical rolling processes. Metallurg, 1959, No. 3

square" system, which ensures that billets of high quality are obtained. Such a mill is simple in construction and yields high working figures.

Taking into account the fact that the lengthwise layout of soaking pits does not satisfactorily ensure the supply of ingots for high production capacity blooming mills or slabbing mills, we suggest a layout of soaking pits in two parallel transverse bays, which will give the possibility of feeding ingots from four bogie-ingot chairs and from four stationary ingot chairs (Figs. 2 and 3). The suggested soaking-pit layout has also been recommended for two-stand two-high reversing cogging mills.

A conference on the working of blooming mills in Soviet factories, held the 7th to 10th December 1954, at which our proposals for the creation of a multi-stand (continuous) blooming mill were discussed, noted the "advanced principle of continuous flow" laid down in this giant blooming mill, determined its production capacity at seven million tons per year or more, and recommended the Ministry of Ferrous Metallurgy of the USSR to commission the planning organizations to undertake the development of the project.

This mill is very well matched with blast furnaces 1700-2700 m^3 in volume, with open-hearth furnaces of 500-800 tons charge, with convertors of 100-270 tons capacity and with high production finished-product mills.

Where it is still possible, the projected buildings for newly planned rolling shops should be re-examined with regard to the installation of continuous blooming mills. They should go into use in large-scale metallurgical factories, and also in factories for special steels.

THE CRIMPING OF STEEL BRUSHES

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To accelerate the removal of scale from the work-piece surface during the pickling of certain steels, a combined method appears the most logical, namely, pickling with mechanical scrubbing.

For this purpose, a scrubbing-washing machine (Fig. 1) has been installed in the pickling equipment: this consists of a welded frame, two alloy steel support rollers, two driven brush cylinders and a spray system, from which water under a pressure of 2.5-3 atm impinges on the work-piece from above and below to wash away the slimy acid solution. The brush cylinders, revolving counter to the motion of the work-piece, are made up from separate discs with metallic brushes. Up to 190-200 individual circular brushes go into the making of one such cylinder. The preparation of the cylinders and dismantling them after wear is a laborious and unproductive operation. Apart from this, the short life of steel brushes made from ordinary wire 0.3 mm thick made it necessary to replace the brush rolls frequently: in its turn this gave rise to lost time on the pickling equipment and reduced its producivity. The service life of the wire in the steel brushes had to be increased.

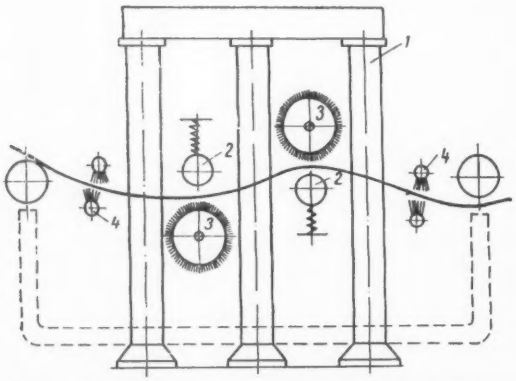

Fig. 1. Scrubbing-washing machine. 1) Frame; 2) support rollers; 3) driven brush cylinders; 4) sprayers.

Workers in the factory suggested crimping the wire in the brushes. Experimental crimping of steel brushes for two cylinders was carried out by hand. Tests on them gave positive results: the life was trebled. In further efforts in the factory, a machine was constructed for crimping wire in steel brushes (Fig. 2). On a welded sheet-metal and angle-iron frame, a worm reduction gear was set up with a gear ratio i = 9.5. On the gear output shaft was rigidly fixed a disk having a radially situated slot in which was fixed a connecting-rod crank-pin.

By moving the connecting-rod crank-pin in the slot, the eccentricity can be altered and at the same time the movement of the connecting-rod can be increased or decreased.

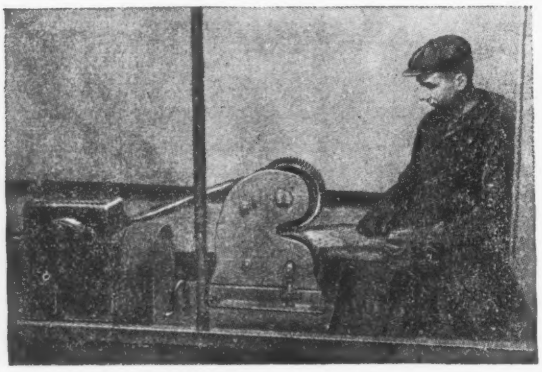

Fig. 2. Crimping machine.

The other end of the connecting-rod is joined to the crimping cog-wheels having a module equal to four, which are driven by the connecting-rod in backward and forward circular motion. The angle of rotation of the wheels, depending on the position of the crank-pin in the disk-slot, can be changed from 30 to 60°.

The distance between the centers of the crimping wheels is altered by means of the lower wheel, the axis of which can be regulated in height.

The machine is driven by an electric motor, 2.7 kw, 1500 rpm, through a Texrope (multiple V-belt) drive with a gear ratio of 2:1 and through the reduction gear described above.

During the reciprocal-circular rotation the cog-wheels draw tufts of the brush into the clearance, crimp it and them push it out. The worker turns the brush over, when it is pushed out from the clearance between the cog-wheels (see Fig. 2).

The use of brushes with crimped wire made it possible to reduce wire consumption by a factor of three; to cut down unplanned stoppages on the pickling equipment and to raise its production capacity; and almost entirely to avoid the heavy manual labor in assembling steel brushes.

IMPROVING THE QUALITY OF SCRAP METAL DISPATCHED

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September, 1960

In striving to complete the tasks for the seven year period before time, Soviet metallurgists are from day to day perfecting the techniques and methods of metallurgical production, and raising the requirements as to the quality of scrap dispatched. The successful solution of this task can be realized only by means of the introduction of advanced methods of pre-treating scrap and waste products of secondary metals.

Definite work has been carried out in this direction by the specialized organizations for the procurement, pre-treatment and dispatch of ferrous and nonferrous scrap of Ukrglavvtormet under the Gosplan (State Planning Committee) of the Ukrainian SSR.

The management areas and the Vtormet factories have been supplemented recently by scrap pre-treatment equipment: there have been installed 40 presses for compacting sheet scrap and turnings, including 3 type PG-400 presses, 5 presses for briquetting cast iron turnings, 26 alligator shears for mechanical cutting and 16 turnings-crushers. At the Dnepropetrovsk Vtorchermet, factory furnaces have been brought into use for pre-heating twisted turnings before pressing.

All this has increased the pre-treatment of ferrous scrap from 1808.5 thousand tons in 1957 to 2255.5 thousand tons in 1959; that is an increase of 24.7%.

As a result of introducing means of mechanization and improving the technique of pre-heating scrap, the compacting of light weight scrap, of steel and cast iron turnings has been increased.

In 1959, 424.3 thousand tons of light weight scrap was compacted as against 323.6 thousand tons in 1957. According to Ukrglavvtormet the average yield of one compacting press in 1959 was 2120 tons, and of alligator shears, 1350 tons. The average weight of a compact was increased from 70 to 110-120 kg, by adding into them up to 40-45% of broken down turnings.

The highest productivity on compacting presses in 1959 was achieved by the following managements: Nikolaevsk–3780 tons; Poltavsk, 3232 tons; Kiev, 3140 tons: on alligator shears, Dnepropetrovsk, 2680 tons, Lugansk, 2440 tons.

Advanced teams of compacting workers by improving the technique have obtained high productivity on compacting presses type B-122. Thus in the Zaporozhsk Vtormet management, a packeting press has been equipped with a stock chamber for charging raw material. This improvement has shortened the charging time in the press chamber to three minutes, which has made it possible for the team of compacting workers to increase production to 12 tons, the norm being 6.4 tons per shift. Similar equipment is used in the Nikolaev, Kiev, and Lugansk managements.

At the Dnepropetrovsk Vtorchermet factory, compacting workers have put into good operation a semi-automatic compacting press PG-400, as a result of which, with a norm for pressing sheet scrap of 59 tons, the team under press-machinist Zaidun, a member of the Young Communist League, compresses up to 70 tons per shift.

A furnace set up in this factory for heating scrap strip up to 12 mm in thickness, as well as cables, wire and spiral turnings before pressing made it possible to compress up to 2000 tons per month of heavy scrap on a mechanical compacting press PV-300.

At the Konstantinovka Vtorchermet factory, a briquetting press type B-653 has gone into operation for briquetting cast iron turnings. The team briquettes up to 7 tons of cast iron turnings per shift. The weight of a single briquette is 4.5-5 kg with a density of 5.5-5.9 kg/ dm^3.

The high production capacity of the briquetting press and the density of the briquette are achieved by means of correct preparation of the turnings, namely, by removing oil and water, drying, handling with a lifting electromagnet and by screening through a vibrating sieve.

In 1959 at this factory 4.1 thousand tons of cast iron turnings were compressed.

The compression of cast iron scrap is an important measure in making the cast iron scrap resources available for use in the iron foundry industry. In the undertakings of the Ukrainian SSR, 160-170 thousand tons of cast iron turnings are formed per year. But the equipment necessary for compacting it is being assembled slowly, and this valuable material is being used at the moment as a metallic addition to the blast furnace charge. The percentage weight of pretreated scrap delivered to shop and management areas rose from 67.5% in 1957 to 74.6% in 1959. In the Zakarpatsk management area it stood at 96.6%, in the Chernovitsii at 89.2% and in the Volynsk at 87.5%.

Ukrglavvtormet plans to equip factories, shops and areas with further scrap pre-treatment apparatus and equipment for mechanization: this will promote an increase in output of goods scrap and improve its quality. Thus at the Konstantinovka factory a second ram will be introduced into production, as well as a powerful hydraulic compacting press type "Zilig" with a production capacity of up to 35 tons per hour and a compact weight of up to 1500 kg.

Construction has started of a powerful ram of production capacity up to 100 thousand tons per year at the Dnepropetrovsk factory. In the third quarter of the year the reconstruction of a ram equipment at the Kharkov factory will be completed.

Together with this, it is possible to bring forward many facts which testify to the untapped reserves for increasing the yield of high quality goods scrap. In the Vtormet shops and management areas it is often possible to discover in the scrap being prepared for dispatch open-hearth steel scrap of class I, II, and III mixed in, as well as electric furnace, alloyed cupola, nonferrous and cast iron scrap. All of this impairs the quality of the metal being smelted.

At the Dzerzhinsk factory, according to the established norm the charging of a 385-ton open-hearth furnace ought to take two hours, but because of light weight uncompacted charge, charging often lasted 3-3.5 hours. It has been calculated that if the charging time in each open-hearth shop could be reduced by only ten minutes, the increase in steel melted would be thousands of tons a month. The position is similar with the metal charge and furnace working at the "Zaporozhstal'," Makeevka factories, etc.

Meanwhile certain workers in the Vtormet system and also directors of undertakings supplying the metal underestimate the importance of compacting and densifying the metal charge dispatched for open-hearth furnaces. For example, the Stalinsk Vtormet management (director, Comrade Nechikov) only during January-February of this year shipped for the Makeevka Metallurgical Factory – the Kirov plant – 9.5 thousand tons of unsorted iron and steel scrap, of which 4.7 thousand tons was light-weight scrap. Such a state of affairs is also found in other Vtormet managements.

Special complaints should be made against machine construction undertakings, in which, as is well known, considerable amounts of steel turnings are produced, but, because of poor collection, storage, and pre-treatment, this valuable raw material for steel melting is turned into a metallic addition for the blast furnace charge.

In the Zaporozh'e region in undertakings subordinate to Sovnarkhoz (Council of National Economy), to Oblispolkom (Regional Executive Committee) and other departments, over 20 thousand tons of steel turnings are formed each year, but in view of the fact that in a number of factories the instructions about collection, storage and pre-treatment of metal scrap are not fulfilled, more than a third of this valuable product is mixed with cast iron turnings and nonmetallic detrimental additions (sand, rubbish and so on) and sent of in the spiral form, ungraded.

The wide introduction of industrial and progressive methods and measures for mechanization in the pre-treatment of ferrous scrap in shops and in management areas and factories of the Vtormet system, as well as in undertakings of metal-suppliers will result in increasing steel scrap resources considerably, and raising its quality.

Translated from Metallurg , 1960, No. 9, pp. 35-36, September, 1960

N. V. Zaberyukha, A. M. Bugeev, L. A. Volkov, and A. A. Bezdenezhnykh, The Casting of Steel in High-Production Open-Hearth Shops (Metallurgizdat, Sverdlovsk, 1960, 216 pages).

In this book, from the material of an interfactory study group, the advanced methods for the casting of steel in large-scale metallurgical undertakings in the USSR (the Magnitogorsk, Kuznetsk and Nizhne Tagil combines, the "Zaporozhstal'," "Azovstal', " Chelyabinsk metallurgical factories and others) are summarized. This summary is supplemented by the authors' investigations and by data from the literature. In some cases the authors give the theoretical basis of a progressive method.

The book consists of seven chapters. In chapter I, "The Steel Ingot, Its Structure and Defects," the problems of the crystallization and structure of ingots of unkilled, killed and semi-killed steel are clarified in a very detailed manner; basic information is presented about the designing of ingots and the classification of their chief defects; the causes for their formation and methods of partially or fully preventing them or eliminating them are considered.

In Chapter II, "Methods and Technology of Casting," the authors describe the improved methods of casting steel used in high-production shops. Bottom-pouring, top-pouring through one or two stoppers, pouring through an intermediate tundish are considered, together with their advantages and disadvantages.

In Chapter III, "Ingot-Molds and Their Accessories," the types of ingot-molds in use, feeder extensions, mold stools, centers, their maintenance and preparation for the melt are distinguished between. The authors, in discussing the question of ingot-mold wall thickness in detail, remark that the majority of factories have not altogether correctly selected the ingot-mold wall thickness in its upper and lower sections. Starting from the incorrect assumption that a thick wall removes a large amount of heat, to secure the principle of directional solidification of killed steel these factories make ingot-molds with thick walls gradually thinning off from the bottom to the top. The authors show that the value of the ingot-mold thickness influences the value of the rate of solidification only in the initial period of the ingot's crystallization and only up to a certain wall thickness. In the later stages of crystallization the amount of heat removed, from the value of the heat loss by radiation and convection from the surface of the ingot-mold, will be greater with a thin ingot-mold wall thickness. However, an excessive reduction in ingot-mold wall thickness leads to a fall in their service life, especially in casting through two stoppers when the ingot-molds are located close to one another on the mold stool.

By comparing ingot-mold dimensions in our factories with data for foreign practice, the authors show that our ingot-molds have a greater wall-thickness: consequently, there is the possibility of reducing the specific consumption of ingot-molds per ton of steel melted.

In Chapter IV, "Steel-Casting Ladles, Their Preparation and Maintenance," the chief types of present day steel casting ladles of larger capacity, their lay-out, the methods used for lining them and the treatment in preparing for each melt are indicated. Much space is devoted to the question of mechanization of work in ladle repair. The layout and maintenance of stoppers and stopper mechanisms are also considered here, together with the use of intermediate ladles (tundishes) and steel pouring-spouts.

In Chapter V, "Casting Bays and Their Equipment," designs of the casting bay building and of the casting floor are described and the working of the railroad transport and crane equipment is considered, together with the main goods traffic in the bay, the main refractory work carried out in the casting bay and their mechanization. Much space is devoted in this chapter to the problems of the rational use of the casting bay floor area. The authors express a number of valuable suggestions about relieving the casting bay of different sorts of subsidiary work: they suggest removing the refractory work, at the moment carried out in the casting bays, into other buildings and organizing special mechanized store-house workshops for repair and storage of reserve equipment.

In Chapter VI, the preparation of rolling stock and ingot-molds for the casting of steel are described. The chief types of bogie being used, and the different methods of installing the ingot-molds on them are shown; the calculation of the necessary amount of rolling stock is carried out; and the question of how long the ingots should be kept in the ingot-molds is considered. The work of the ingot-mold yard, of the stripping and spraying sections are described, as well as the sections for cleaning and coating the ingot-molds. A special section in this chapter is devoted to the organization of rolling stock preparation on a continuous basis, and to the mechanization of the individual operations.

In the final chapter, Chapter VII, "Problems in the Organization and Creation of Safe Working Conditions," problems are discussed concerned with the rational organization of labor in casting bays, ingot-mold yards, and in stripping sections: these are very important for increasing casting bay output capacity. Special attention is paid to the tables of organization and arrangement of workers. Certain measures are also discussed here for creating safe working conditions in the casting of steel and the preparation of rolling stock.

No objection can be raised against the majority of problems discussed by the authors. However, certain of the book's deficiencies should be pointed out.

In discussing the phenomena taking place during crystallization of the ingot, the different points of view held by metallurgists as to their nature should have been clarified – even if only briefly – and the reader referred to the appropriate literature sources for a more complete account.

In considering keeping the feeding parts of the ingot hot the authors should have dwelt at greater length on the methods widely used (the use of refractory hot-tops and covering the metal surface with a cap), and just because of this it is necessary to undertake an account of this section. It would have been useful to discuss problems of the shape of the feeder head and methods of reducing its capacity to stove heat, as well as the problems of the preparation of caps. There are no grounds for the authors statement that the method of supplementary electric heating is "comparatively simple" and could be used for high production open-hearth shops.

It is doubtful whether the ad hoc suggestions of the authors about lining the hot-top in two layers – an internal layer next to the liquid metal of fireclay brick laid flat and a cope reinforcement – can be adopted. Even if such a lining is more sensible from the point of view of service life, from the point of view of the thermal working of the feeder head it is less effective than a lining made up of one brick laid flat, since in the case of a thicker extension-piece lining the amount of heat lost from the feeder head metal in heating the head up is increased.

The authors correctly remark that it is inadmissable to feed killed steel ingots to the stripping section and to remove the hot-tops from them before they have completely solidified. However, in conclusion the authors state that with the aim of increasing ingot-mold life, increasing the turn-round of rolling stock and so on, the time ingots remain in the molds should be reduced. In any case ingots should be kept in the molds for the minimum time to obtain a sound ingot. If, in this case, the turn-round of rolling stock is inadequate, then it should be increased by other measures, not by bringing down the production quality.

In describing the construction of the casting bay building, several times letter designations for rows of columns are given, but neither in the section of the casting bay (Fig. 60) nor in the plan (Fig. 61) are they shown; moreover, in this subsection, there are not, in general, references even to the figures which are in the book. In the same section the authors state, not entirely accurately, that the distance between columns in the furnace bay is 30-37 m. It is well known that the distance between columns in shops is always a multiple of six, and therefore 30-36 m should have been stated.

In certain formulas set out in the book symbols are inaccurately given.

Finally, the authors do not always successfully treat certain technical terms. Thus the row of refractory in the ladle, lying next to the casing, is called on one occasion the "reinforcement" (which is correct), on another the "sheathing," on another the "master-lining"; the part of the ladle lining onto which the first portions of metal fall in pouring is called the "combat part of the ladle." The terms introduced by the authors are, in my opinion, unfortunate: "thick ingot" and "ratio of the ingot height to its thickness" and then also "ratio of ingot height to its breadth."

In the book, unfortunately constructed phrases are encountered. The authors use a number of local turns of speech and expressions. There are a number of annoying misprints of an important character.

The points mentioned should be corrected in subsequent editions.

In its aim, the edition of the book under review is justified and timely. No doubt the book will be of practical advantage to workers in steel-melting shops in eliminating those technical and organizational difficulties which arise in casting steel in present day high-production open-hearth shops.

Engineer L. A. Smirnov. Urals Scientific Research Institute for Ferrous Metals.

The Production of Railroad Wheels, Tires, and Axles Abroad. A collection of articles under the editorship of Professor Dr. Mech. Sci. A. P. Kul'bachnii. (Moscow, Metallurgizdat, 1960, 208 pages).

Reports on the production of railroad axles, tires and wheels in many countries are collected in this book. The first section is devoted to experience in their preparation in the USA, England, the Federal German Republic, Czechoslovakia and the German Democratic Republic.

In the second section the treatment of these details is considered.

In the third section reports about methods of investigation are collected, and in the fourth, information is given about the construction, standard specifications and defects of wheels, axles and tires.

The book can be useful to engineer-technicians and scientific workers in ferrous metallurgy.

USA STEEL PRODUCTION

PROCEEDINGS OF THE 41ST CONFERENCE OF STEEL PRODUCERS OF THE USA* (Open Hearth Proceedings, 1958)

Translated from Metallurg, 1960, No. 9, pp. 37-40,
September, 1960

A characteristic feature of the growth of metallurgy in recent years is the ever-increasing proportion of steel melting by countries of the socialist camp and the reduced share of the USA in world steel production, which are, in percentages:

	1950	1955	1957	1958
Countries of the socialist camp	19.1	23.3	25.3	31.1
USSR	14.4	16.8	17.6	20.1
USA	–	40.0	35.0	30.1

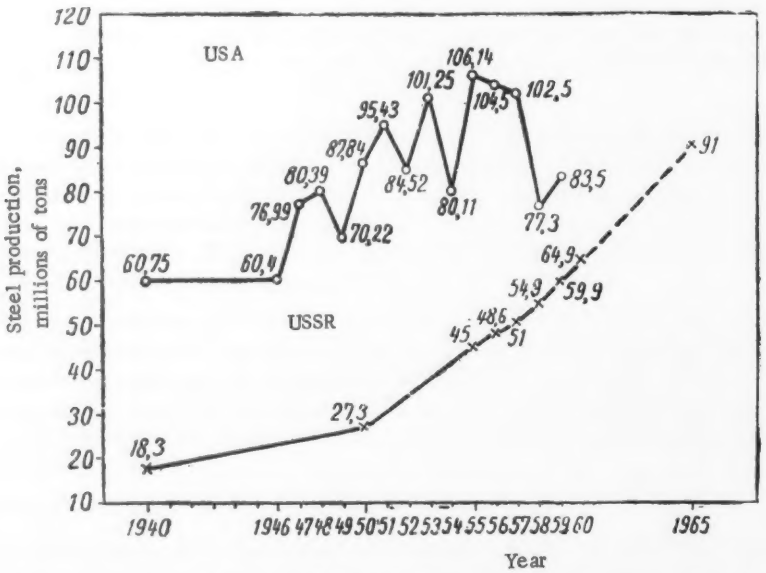

Fig. 1. Dynamics of the growth of steel in the USSR and the USA, in millions of tons.

* The 43rd Conference of Steel Producers was held in Chicago in 1960 but only the proceedings of the 41st Conference have been received in the USSR up to the present time.

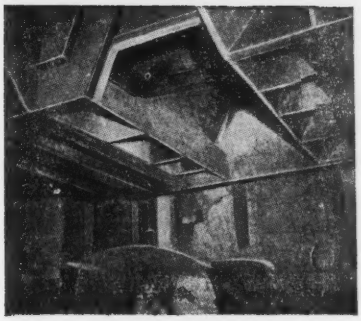

Fig. 2. Opening in the botton of hopper for scrap over cart with molds for their loading.

In 1959 world steel production was about 300 million tons, which is the maximum during all the years of growth of ferrous metallurgy (in spite of a considerable drop in steel production in the USA). The dynamics of the growth of steel production in the USSR and the USA by years are shown in Figure 1.

The Proceedings of the 41st Conference on Steel Melting Production, which occurred in Cleveland, Ohio, USA, has served as the source for a comparison of technical directions in the steel melting production of the USSR and the USA.

The Conference on Steel Production in the USA has been called by the National Committee annually, since 1925; metallurgists, physicists, chemists and practical men of various enterprises and institutes are attracted to these conferences. At the 41st Conference there were 1146 specialists present, from various sections of the United States. Papers on different subjects have been collected in the Proceedings of this Conference.

A long report by L. E. Reinartz was devoted to a review of the development of the various types of steel production during the last hundred years.

Basic attention was given in the report to the development of the process of melting steel in basic converters with oxygen supplied from above (the L-D process) and the use of this process in Europe and the United States.

Having touched upon the "Cal-Do" and rotary processes used in Europe, the speaker suggested that those processes will also be used in the United States sometime for the removal of silicon and carbon from cast iron in order to speed up the open hearth or electric furnace processes.

Interesting information is given in the same Proceedings on the technical thermodynamics of the process in the oxygen converter. A report is devoted to calculation of the material balance of smelting done with a charge containing 40-45% scrap, 50-52% open hearth cast iron and up to 5% ore. It is reported that the quantity of scrap can be brought up to 50%.

At the session of the blast furnace technology section there was discussion of problems in the organization of the charging of materials, the use of hoists for scrap during the charging of open hearth furnaces, automation of control of the weight of open hearth melt, acceleration of the charging cycle, improvement in the technology of steel melting (rimming type), the use of exothermic ferromanganese and durability of molds for the production of wide slabs.

In connection with the fact that while scrap is being charged into large open hearth furnaces with molds the carts occupy a great deal of space on the tracks (about 10.3 meters), block off neighboring furnaces and hinder the aftercharging of limestone and ore, the Armco Company at its Butler Yorks plant has applied a new method of efficient utilization of railroad tracks in the working areas. Scrap in the working area is put in special cars, each of which replaces eight to ten cars for molds used earlier.

A crane with a suspended magnet supplies the necessary portion of scrap to a Calderon pouring hopper located on a cart with three molds; then a mold is put in the open hearth furnace with an ordinary charging machine and the magnet loads another portion of scrap into the hopper.

This studies have shown that the turnover of the molds is increased proportionally. The time spent in loading a large cart with scrap in the charging yard is 15% less than in loading molds.

A large section of the Proceedings was devoted to the use and service of refractories. Recently instead of brick 230x113x65 mm a thicker brick with dimensions of 230x113x76 mm is being used more and more extensively. An overwhelming majority of the refractory brick is burned after shaping and drying. Some types of basic brick are combined chemically and are not burned before use.

At the present time some furnaces in the United States are operating excellently with heads lined with basic brick. In this the entire end of the furnace including the crown of the heads from a point on the flame

altar, the end wall, the side wall, the vertical channels and at least the upper part of the slag chamber, are faced with basic brick.

The basic lining of the head crowns in vertical channels is as a rule suspended to eliminate excessive tensions in those sections. That design of crown has been borrowed from the USSR. Using the experience of the USSR, they are beginning to use unburned reinforced chrome-magnesites in the United States.

In closing, the author concludes that in comparing the conditions of use of flat and arched crowns it is difficult to give preference to one or the other design of crown.

It is considered that the use of lithium refractories leads to increase in the durability of crowns, increase in steel smelting and considerable reduction of its cost.*

Methods of laying and repairing bottoms are described in the Proceedings. The bottom of an open hearth furnace requires annual care, since cavities are formed in it due to the removal of slag, or humps develop, formed by limestone which sometimes remains on the bottom after the smelt is poured off.

Slag remaining on the bottom hardens when the furnace is charged with cold scrap, and its surface can be covered with limestone charges. After the metal has been poured off and the limestone comes to the surface over the slag, the liquid metal comes in contact with the surface of the solid, but the slag containing ferric oxide floats free. These oxides begin to react with carbon of the bath and cause local circulation of metal, which at first absorbs the liquefied slag and then mechanically scours the low-lying flat surface of the bottom.

When the bottom is poorly made the steel can flow into cracks or voids and, hardening there during the time of subsequent charging of the furnace, will act as a center for the evolution of gas.

Recently they have begun to use a bottom lined with carbon blocks which according to the reporter's conclusion enables them to reduce down time of the furnace for repair of the bottom by 60%. After repair of the furnace such a bottom dries much more rapidly and consequently the time of repair is reduced.

A bottom of carbon blocks has been made for a 425-ton open hearth furnace in Cleveland. It is 16-17% more costly than an ordinary lining, but labor costs are 20-25% lower.

Participants in the conference directed much of their attention to problems in pouring steel, the making of molds and the use of hot tops.

In the USSR it has always been considered that steel pouring and the use of molds should be considered highly specialized and responsible segments in the work of steel mills.

In a report by Robert A. Benting it was pointed out that at the Lukens Steel Plant about 4000 ingots per month are cast in molds with risers.

Ingots weighing 1.3-7 tons (2900-15000 pounds are cast in molds of 15 different types: the smallest has dimensions of 812x305x1824 mm (32x12x60 inches) and the largest 2745x1016x3124 mm (108x40x123 inches). Such an abundance of mold types makes the manufacture of hot tops, as well as the procurement of casting supplies, a complex problem. In connection with this the possibility has been studied of producing molds with the organization of a special plant for the production of hot tops. This made it possible to satisfy the plant

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* The upper structure of the majority of open hearth furnaces in the USSR is completely lined with high-refractory and slag resistant materials (94.2% of all the open hearth furnaces operating in ferrous metallurgy, producing 98.4% of all the steel, have basic crowns).

Comparative tests conducted by the Khar'kov Institute of Refractories jointly with the "Azovstal' " plant on the durability of smooth crowns 380 and 460 mm thick have shown that the durability of crowns lined with brick 460 mm thick is somewhat higher but not directly proportional to the increased thickness of the crowns.

Study of the operation and durability of open hearth furnaces indicates the need for a return to the practice of completing thrust-suspension crowns with a working area that is ribbed and not smooth.

New types of crown brick have been developed in the USSR for the improved construction of the main crown, thermal insulation of the main crown and the supporting arches of vertical channels.

requirements in riser extensions. There is distribution of hot tops made completely of refractory clay; hot tops of a second type are relatively steady - they consist of a cast iron or steel framework lined with refractory brick which has withstood more than 100 melts at some plants. Between melts the surface of the brick lining is usually touched up with a suspension of refractive clay.

This report also contains information on the use of riser extensions of refractory cement, consisting of a very pure quartz sand (95% SiO_2), fresh Portland cement and reinforcing wire with a 6.5 mm diameter. The recommended sand particle sizes are: 5-15% of 20-30 mesh, 8-16% of 100 mesh or smaller and the remainder of 40-70 mesh.

The sand and cement are mixed in a ratio of 7:1 and the mixture charged into the crushing mill of a special charging machine and reground for one minute, and then water is added amounting to 5-6.5% by weight of the mixture.

The complete cycle from load to load takes five minutes.

A communication on work done in the automation of the melting process is of interest. A study of several thousand ingots of the same weight cast by the ordinary method has shown that individual ingots vary in weight by ± 8%. These variations have several causes: 1) the melter does not observe the metal during filling of the mold because of smoke and flame in the presence of burning grease; 2) steels behave differently during cooling and the evolution of gases; 3) worn molds have large volume.

The melting of ingots with the same weight is done with the "Ingotrol" automatic system, consisting of weights suspended on a hook, operating precisely under high temperature conditions and enabling given portions of metal to be weighed off. The operating principle of this system is that the weighing device on which a ladle with the metal is suspended gives an impulse for automatic closing of the stopper during the melting of a given, predetermined amount of metal.

Large-scale industrial testing of the system has shown that in pouring ingots with the same weight variations in the weight of ingots amounts to ± 1%.

At the 41st Conference of Steel Producers of the USA light was also thrown upon theoretical questions connected with the thermodynamics and kinetics of metallurgical processes, the physicochemical nature of metallurgical slags, methods of control and regulation of metallurgical processes, discussion of the processes of diffusion and distribution of elements in metals and alloys and the processes of desulfuration and dephosphoration.

Steel production in the USSR is developing in the direction of the enlargement of steel smelting aggregates, improvements in the design and technology of smelting, the intensification of the process with use of oxygen and basic refractory materials, the automation of processes and the mechanization of laborious work.

The USSR in recent years has considerably surpassed American indices both as to the rate and as to the absolute dimensions of growth in steel melting, as is evident from the following data:

	USSR	USA
Average annual rate of growth of steel melting 1954-1959, %	7.8	−2.9
Average annual absolute growth in steel melting 1954-1959, millions of tons	3.6	−2.7

The level of absolute growth of steel melting in the USSR is large, and the rate of growth is so much more rapid that the time is not distant when the USSR will overtake and pass the USA in steel melting and in the total volume of industrial production.

D. A. Smolyarenko, Candidate in Technical Sciences, Chairman of the All-Union Steel Melting Section of the Scientific Technical Department of Ferrous Metallurgy.

METALLURGY PRESS WILL PUBLISH IN THE FOURTH QUARTER OF 1960:

V. A. Verkhushkin, Technical Normalization of Metalworking Production. Ten quires, price 6r. 50k. Practical manual for engineers, economists and standardizers.

V. A. Efimov, Steel Melting and Ingot Formation. 21 quires, price 12 r. For engineering-technical workers of the metallurgical and machinery industries.

N. K. Leonidov, Improvement in the Design of Blast Furnaces. Five quires, price 4r. For engineering-technical workers of the metallurgical and machinery industries.

K. Ékhel'khoizer (translation from German), The Operation of Industrial Furnaces Fueled by Gas. 8.6 quires, price 6 r. For factory workers engaged in the operation of furnaces; can be useful to designers planning industrial furnaces.

Price and sizes of books are approximate. Order books you need in advance in local bookstores of the Book Trade Administration. In case of refusal to accept order, communicate with: Metallurgy Press, Sales Department, 2nd Obydenskii per., 14, Moscow G-34.

Textbooks and Educational Reference Books on Sale in Bookstores of the Book Trade Administration

N. V. Amchislavskii, The Metal Cutter in Rolling Mills. Textbook for the production and technical instruction of workers. 1957, price 1 r. 90 k.

F. I. Beland, Water Preparation. Collection of problems. Educational reference book for technical schools of the metallurgical industry. 1958, price 8 r. 60 k.

P. T. Emel'yanenko, A. A. Shevchenko and S. I. Borisov, Pipe Rolling and Pipe Welding Production. Textbook for schools and courses for foremen. 1958, price 11r. 60 k.

V. F. Ivan'ko, The Control Panel of an Arc Steel Melting Furnace. Textbook for the production and technical instruction of workers. 1957, price 3 r. 15 k.

D. G. Ignatenko, A. L. Starosel'skii et al., Machinist-Operator of a Rolling Mill Control Post. Educational reference book for the production and technical instruction of workers. 1957, price 4 r. 60 k.

M. L. Mirenskii, Work on a Light Section Rolling Mill. Educational reference book for schools and courses for foremen. 1957, price 6 r. 50 k.

B. Natapov, Metal Working. Educational reference book for ferrous metallurgy technical schools. 1956, price 8 r. 35 k.

A. P. Pankevich, Handbook of Methods for a Course in Drawing. Educational reference for metallurgical technical schools. 1958, price, 35 r. 90 k.

M. I. Panfilov, Steel Production in Open Hearth Furnaces. Educational reference book for schools and classes for foremen . 1957, price 9 r. 95 k.

G. S. Romashkovtsev, The Controller of an OTK Open Hearth Furnace. Educational reference book for technical schools. Can be suitable for workers in OTK steel mills of metallurgical and machinery plants. 1959, price, 4 r. 30 k.

V. L. Timofeev and M. K. Tkachenko, The Production of Open Hearth Steel. Textbook for schools and courses for foremen. 1957, price, 5 r. 15 k.

Yu. M. Chizhikov, Rolling Mill Production, 2nd revised and enlarged edition. Textbook for metallurgical technical schools. 1958, price, 14 r. 45 k.

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Translated from Metallurg, No. 9, inside back cover, September, 1960

Problems of Kachkanar. From the materials of the conference on the large-scale use of the Kachkanar titanomagnetite ores; 187 pages Sverdlovsk, 1959. Published by the Central Office of Scientific Research Information of the Sverdlosk Council of National Economy.

The collection contains reports and abridged speeches of the participants of the scientific and production conference on the large-scale utilization of the Kachkanar titanomagnetite ores which was held at Nizhne Tagil in 1958. The material in the collection sheds light on the degree of knowledge and reserves of the Kachkanar titanomagnetite deposits, the progressive methods of their extraction, concentration, and metallurgical processing, the technical and economic indices of the utilization of the Kachkanar ores, and also on the progress of the construction of the priority mining-concentration combine.

V. I. Dovgopol's report, "Large-Scale Development of the Tagil-Kushvinsk Industrial Region," sheds light on the prospects of developing ferrous and nonferrous metallurgy in the region and on the problems of providing the enterprises with an ore base – the construction of the Kachkanar Mining-Concentration Combine and the Volkovsk copper-iron mine.

N. G Tyurenkov's report "Concentration of Kachkanar Ores" cites the results of the experiments on concentrating two types of Kachkanar ores (weathered and bedrock ores) under industrial conditions at the experimental concentration factory at the Krasnoyarsk plant "Sibelektrostal'"; results are given on the concentration of ores by different technological methods and a conclusion is reached concerning the advantage of using the method of wet magnetic stage concentration with dry magnetic separation at the head of the process in order to obtain a high-quality iron-vanadium concentrate.

The report by G. I. Sladkov, "A Plan for Concentration and Sinter Factories," contains a description of the plan for the factories of the Kachkanar Combine which was worked out by the Ural Mechanical Engineering Office. The plan provides for concentration of the ore according to a method which includes four-stage crushing of the ore in an open cycle, dry magnetic separation, pulverizing in open and closed cycles and wet magnetic separation. The finished iron concentrate is sent to the sinter factory. The report examines the advantages of this method of concentrating the Kachkanar ore in comparison with concentrating the finely disseminated magnetite-quartzites of Krivoi Rog and the Kursk Magnetic Anomaly, as well as the taconites of Silver Bay and Erie (USA)- the productivity and composition, equipment and design of the framework of the factories are given.

In the report "The Metallurgical Value of the Finely Pulverized Kachkanar Concentrates, the Technology of Sintering Them, and Features of Blast-Furnace Smelting" by N. M. Babushkin, V. A. Shamarin, D. G. Khokhlov, S. A. Yel'kin, and L. Ya. Gavrilov are given the particulars of using the Kachkanar titanomagnetites in metallurgical units and the conclusion is made concerning its value for obtaining vanadium pig iron (thanks to the high basicity of the barren rocks, purity with respect to sulfur and phosphorus, the presence of vanadium and good ability to be concentrated); results are cited of the laboratory investigations on sintering the finely pulverized Kachkanar concentrates by the methods of sintering and nodulizing with subsequent roasting of the pellets, wherein the advantage is noted of the latter method over sintering. Data on the industrial tests of fluxing sinter are given.

The report of A. I. Pastukhov and M. A. Tret'yakov, "Metallurgical Processing of Pig Iron from the Kachkanar Titanomagnetite into Steel," is devoted to an examination of the methods of converting vanadium pig iron with the obtainment of vanadium slag suitable for producing ferrovanadium. It is pointed out that the most probable method of conversion is by blowing through the pig iron in a converter. The results are given of industrial tests of blowing through the Kachkanar pig iron with a bottom air blast in a converter. After separation of the vanadium slag from the semi-finished product it is possible to smelt the steel in an oxygen converter, rotary electric steel furnace, or in an open-hearth furnace.

M. V. Levin's report, "The Use of Concentration Wastes of the Kachkanar Titanomagnetites for the Production of Construction Materials" gives a detailed qualitative evaluation of construction material obtainable from the concentration of the Kachkanar ores; results are cited of the experiments of using rock dust in construction and points out the need for solving the problem of using the dustlike wastes of the combine.

The report of A. A. Rubinshtein, "The Development of the Production of Construction Materials from the Waste of the Kachkanar Mining-Concentration Combine," is devoted to the problems of processing the wastes of the combine: stripped rocks, oreless wastes, dry finely crushed wastes of the crushing factory, and tailings of the wet magnetic separation process.

The report by A. S. Osintsev, "The Economics of the Large-Scale Use of the Kachkanar Titanomagnetites," investigates the problems of the cost of production of the rock materials from the stripped rocks and concentration wastes of the Kachkanar Mining-Concentration Combine and certain rubble stone and ballast rock quarries of the USSR; an economic estimate is made of the production of iron-vanadium concentrate from the Kachkanar titanomagnetites according to a concentration method which includes dry and two-stage wet magnetic separation with crushing of the ore to 20-0 mm and pulverization of the intermediate product in the first stage to 1.5-0 mm and in the second stage to 0.21 mm, as well as calculations of the cost of the Kachkanar agglomerate; a comparison is made of the main technical and economic indices of the exploitation of the Kachkanar deposit and of other iron-ore basins of the USSR; problems of the economics of the metallurgical processing of the iron-vanadium agglomerate are cited.

During the discussions, the participants of the conference indicated the need for a prior prospecting of the Kachkanar titanomagnetites and for the construction at the deposit of an experimental concentration factory or an ore testing station. Comments were made on the plans of the concentration and sintering factories and also on the problems of developing a more complete method of preparing the ores for blast-furnace smelting, which should be based on the second-priority planning of the combine.

The conference noted the need to put the Kachkanar deposit of titanomagnetites into operation as rapidly as possible, and resolved to conduct further geological explorations of the Kachkanar and Gusevogorsk deposits; to conduct investigations of the Kachkanar ores for the purpose of clarifying the possibility of obtaining from them not only iron-vanadium concentrates but also other concentrates; to continue experimental investigations on the preparation of Kachkanar concentrates for blast-furnace smelting and on the development of an efficient method of processing the Kachkanar vanadium-pig iron; to include in the composition of the Kachkanar Mining Concentration Combine the construction of enterprises producing construction materials based on the wastes of the mining and concentration production.

SIGNIFICANCE OF ABBREVIATIONS MOST FREQUENTLY ENCOUNTERED IN SOVIET PERIODICALS

FIAN	Phys. Inst. Acad. Sci. USSR.
GDI	Water Power Inst.
GITI	State Sci.-Tech. Press
GITTL	State Tech. and Theor. Lit. Press
GONTI	State United Sci.-Tech. Press
Gosénergoizdat	State Power Engr. Press
Goskhimizdat	State Chem. Press
GOST	All-Union State Standard
GTTI	State Tech. and Theor. Lit. Press
IL	Foreign Lit. Press
ISN (Izd. Sov. Nauk)	Soviet Science Press
Izd. AN SSSR	Acad. Sci. USSR Press
Izd. MGU	Moscow State Univ. Press
LÉIIZhT	Leningrad Power Inst. of Railroad Engineering
LÉT	Leningrad Elec. Engr. School
LÉTI	Leningrad Electrotechnical Inst.
LÉTIIZhT	Leningrad Electrical Engineering Research Inst. of Railroad Engr.
Mashgiz	State Sci.-Tech. Press for Machine Construction Lit.
MÉP	Ministry of Electrotechnical Industry
MÉS	Ministry of Electrical Power Plants
MÉSÉP	Ministry of Electrical Power Plants and the Electrical Industry
MGU	Moscow State Univ.
MKhTi	Moscow Inst. Chem. Tech.
MOPI	Moscow Regional Pedagogical Inst.
MSP	Ministry of Industrial Construction
NIIZVUKSZAPIOI	Scientific Research Inst. of Sound Recording
NIKFI	Sci. Inst. of Modern Motion Picture Photography
ONTI	United Sci.-Tech. Press
OTI	Division of Technical Information
OTN	Div. Tech. Sci.
Stroiizdat	Construction Press
TOÉ	Association of Power Engineers
TsKTI	Central Research Inst. for Boilers and Turbines
TsNIÉL	Central Scientific Research Elec. Engr. Lab.
TsNIÉL-MÉS	Central Scientific Research Elec. Engr. Lab.-Ministry of Electric Power Plants
TsVTI	Central Office of Economic Information
UF	Ural Branch
VIÉSKh	All-Union Inst. of Rural Elec. Power Stations
VNIIM	All-Union Scientific Research Inst. of Meteorology
VNIIZhDT	All-Union Scientific Research Inst. of Railroad Engineering
VTI	All-Union Thermotech. Inst.
VZÉI	All-Union Power Correspondence Inst.

Note: Abbreviations not on this list and not explained in the translation have been transliterated, no further information about their significance being available to us - Publisher.